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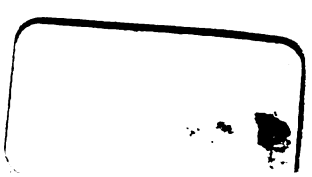

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TRANSACTIONS

OF THE

AMERICAN INSTITUTE OF MINING ENGINEERS.

VOL. XXXVIII.

CONTAINING THE PAPERS AND DISCUSSIONS OF 1907.

NEW YORK, N. Y.:
PUBLISHED BY THE INSTITUTE,
AT THE OFFICE OF THE SECRETARY.
1908.

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WILLIAM GEORGE NEILSON.

See Biographical Notice, p. 402.

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PREFACE.

THIS volume comprises all the proceedings, papers, and discussions of the Institute published during 1907, with the following exceptions:

1. Brief obituary notices of Members and Associates reported as deceased during 1906.[1]

2. Illustrated description of the United Engineering Society Building,[2] and of the exercises attending its dedication, including addresses by Andrew Carnegie, Arthur T. Hadley, Samuel Sheldon, F. R. Hutton, John Hays Hammond, and T. Commerford Martin.[3]

3. Library reports and other announcements of general but temporary interest, furnished to members in the *Bi-Monthly Bulletin*, Nos. 13 to 18, during the year 1907.

4. Address of welcome at the New York Meeting, April, 1907, by Dr. R. W. Raymond, Secretary.[4]

5. Account of the excursion to Cobalt and Sudbury, and entertainments in connection with the Toronto Meeting, July, 1907.[5]

6. Copy of the Report of the Standardization Committees of the Institution of Mining and Metallurgy: Weights and Measures, and Mesh.[6]

7. Year Book, containing the List of Members and Associates revised to Jan. 1, 1908, pp. 173. Published in separate form and distributed with *Bi-Monthly Bulletin*, No. 19, January, 1908.

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[1] *Bi-Monthly Bulletin*, No. 16, July, 1907, pp. 657 to 688.
[2] *Bi-Monthly Bulletin*, No. 15, May, 1907, pp. vi to xxvi.
[3] *Bi-Monthly Bulletin*, No. 15, May, 1907, pp. xxxviii to lxv.
[4] *Bi-Monthly Bulletin*, No. 15, May, 1907, pp. 543 to 547.
[5] *Bi-Monthly Bulletin*, No. 17, September, 1907, pp. 855 to 875.
[6] *Bi-Monthly Bulletin*, No. 14, March, 1907, pp. xxxix to xli.

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The publications, thus mentioned above, exceed 400 pages of valuable material, which could not be included in this volume of the *Transactions* on account of lack of space.

On the other hand, this volume includes certain discussions referring to papers contained in it, which were received after Jan. 1, 1908, yet in time to be here printed instead of being held over for Volume XXXIX.

JOSEPH STRUTHERS,
Assistant Secretary and Editor.

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For the year ending February, 1908.

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JOHN HAYS HAMMOND.....NEW YORK, N. Y.
(Term expires February, 1908.)

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J. B. GRANT.....DENVER, COLO.
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(Term expires February, 1908.)
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JOHN A. CHURCH.....NEW YORK, N. Y.
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(Term expires February, 1909.)

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(Term expires February, 1908.)
THEODORE DWIGHT.....NEW YORK, N. Y.
WALTER WOOD.....PHILADELPHIA, PA.
WM. FLEET ROBERTSON.....VICTORIA, B. C., CAN.
(Term expires February, 1909.)
B. F. FACKENTHAL, JR.....EASTON, PA.
H. O. HOFMAN.....BOSTON, MASS.
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(Term expires February, 1910.)

SECRETARY OF THE COUNCIL.

R. W. RAYMOND.....29 W. 39th St., NEW YORK, N. Y.
(Term expires February, 1908.)

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JOSEPH STRUTHERS.....NEW YORK, N. Y.

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R. W. RAYMOND, Secretary; FRANK LYMAN, Treasurer;
JOSEPH STRUTHERS, Assistant Secretary and Assistant Treasurer.

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JAMES DOUGLAS, JAMES F. KEMP, ALBERT R. LEDOUX.
(Term expires February, 1908.)

THEODORE DWIGHT, CHARLES H. SNOW, R. W. RAYMOND.
(Term expires February, 1909.)

JAMES GAYLEY, CHARLES KIRCHHOFF, FRANK LYMAN.
(Term expires February, 1910.)

Consulting Attorneys, Blair & Rudd, New York, N. Y.

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* SECRETARY'S NOTE.—The Council is the professional body, having charge of the election of members, the holding of meetings (except business meetings), and the publication of papers, proceedings, etc. The Board of Directors is the body legally responsible for the business management of the Corporation, and is therefore, for convenience, composed of members residing in New York.

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POSTSCRIPT.

The list of officers on the preceding page is that of the year 1907, which is the period covered by the contents of this volume. But the result of the election held at the Annual Meeting, February, 1908, although strictly belonging to the next volume, is here published for the convenience of members.

The following officers were elected at the Annual Meeting, Feb. 18, 1908:

COUNCIL.

PRESIDENT.

(*To serve for one year. Term expires February*, 1909.)

JOHN HAYS HAMMOND, ... New York, N. Y.

VICE-PRESIDENTS.

(*To serve for two years. Term expires February*, 1910.)

J. PARKE CHANNING, ... New York, N. Y.
F. W. DENTON, ... Painesdale, Mich.
JOHN B. FARISH, ... Denver, Colo.

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(*To serve for three years. Term expires February*, 1911.)

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R. V. NORRIS, ... Wilkes-Barre, Pa.
W. H. SHOCKLEY, ... Tonopah, Nev.

SECRETARY.

(*To serve for one year. Term expires February*, 1909.)

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ASSISTANT SECRETARY AND EDITOR (BY APPOINTMENT).

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(*To serve for three years. Term expires February*, 1911.)

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ALBERT R. LEDOUX, ... New York, N. Y.

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PAST OFFICERS.

PRESIDENTS.

*DAVID THOMAS	1871
R. W. RAYMOND	1872–1874
*A. L. HOLLEY	1875
*ABRAM S. HEWITT	1876
*T. STERRY HUNT	1877
*ECKLEY B. COXE	1878–1879
*WILLIAM P. SHINN	1880
WILLIAM METCALF	1881
*RICHARD P. ROTHWELL	1882
ROBERT W. HUNT	1883
JAMES C. BAYLES	1884–1885
ROBERT H. RICHARDS	1886
*THOMAS EGLESTON	1887
WILLIAM B. POTTER	1888
RICHARD PEARCE	1889
*ABRAM S. HEWITT	1890
JOHN BIRKINBINE	1891–1892
H. M. HOWE	1893
JOHN FRITZ	1894
*J. D. WEEKS	1895
E. G. SPILSBURY	1896
*THOMAS M. DROWN	1897
C. KIRCHHOFF	1898
JAMES DOUGLAS	1899–1900
E. E. OLCOTT	1901–1902
ALBERT R. LEDOUX	1903–1904
JAMES GAYLEY (Council)	1905
JAMES GAYLEY (Corporation)	1905 ——
ROBERT W. HUNT (Council)	1906
JOHN HAYS HAMMOND (Council)	1907 ——

SECRETARIES.

*MARTIN CORYELL	1871–1872
*THOMAS M. DROWN	1873–1884
R. W. RAYMOND	1884 ——

TREASURERS.

J. PRYOR WILLIAMSON	1871–1872
*THEODORE D. RAND	1872–1903
FRANK LYMAN	1903 ——

* Deceased.

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HONORARY MEMBERS.

PROF. RICHARD ÅKERMAN.....Stockholm, Sweden.
ANDREW CARNEGIE.....New York, N. Y.
DR. JAMES DOUGLAS.....New York, N. Y.
PROF. HATON DE LA GOUPILLIÈRE.....Paris, France.
R. A. HADFIELD.....London, England.
PROF. HANS HOEFER.....Leoben, Austria.
PROF. HENRI LOUIS LE CHATELIER.....Paris, France.
M. FLORIS OSMOND.....Paris, France.
JOHN E. STEAD.....Middlesbrough, England.
PROF. DIMITRY CONSTANTIN TSCHERNOFF.....St. Petersburg, Russia.
CHARLES D. WALCOTT.....Washington, D. C.
PROF. DR. HERMANN WEDDING.....Berlin, Germany.

HONORARY MEMBERS (*Deceased*).

BELL, SIR LOWTHIAN.....1904
CASTILLO, A. DEL.....1895
CONTRERAS, MANUEL MARIA.....1902
DAUBRÉE, A.....1896
DROWN, THOMAS M.....1904
GAETZSCHMANN, MORITZ.....1895
GRUNER, L.....1883
HUNT, T. STERRY.....1892
KERL, BRUNO.....1905
LE CONTE, JOSEPH.....1901
LESLEY, J. P.....1896
PATERA, ADOLPH.....1890
PERCY, JOHN.....1889
POSEPNY, FRANZ.....1895
RICHTER, THEODOR.....1898
ROBERTS-AUSTEN, W. C.....1902
SERLO, ALBERT.....1898
SIEMENS, C. WILLIAMS.....1883
THOMAS, DAVID.....1882
TUNNER, PETER R. VON.....1897

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* Annual meeting for the election of officers. The rules were amended at the Chattanooga meeting, May, 1878, changing the annual election from May to February.

† Begun in May at Easton, Pa., for the election of officers, and adjourned to Philadelphia.

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XCI.	London, England	July, 1906	xxxvii.	xlviii.
XCII.	New York, N. Y.	April, 1907	xxxviii.	lii.
XCIII.	Toronto, Canada	July, 1907	xxxviii.	lix.

* Annual meeting for the election of officers.

† Begun in February at New York City, for the election of officers, and adjourned to Florida.

‡ " " " " " " " " " " " " " " to Philadelphia.

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PUBLICATIONS.

THE publications of the Institute comprise:

TRANSACTIONS.

The volumes of *Transactions*, which are published annually, contain the list of officers, rules, etc., the Proceedings, and the papers revised for final publication. (In this revision, after the preliminary publication, authors are permitted to use the largest liberty; and the changes and additions made in papers are sometimes important. It should be borne in mind by those who study or quote a paper in the preliminary edition, that they may not have in that form the ultimate and deliberate expression of the author's views. It should be added, however, that in the majority of cases there are no important changes.) These volumes are for sale as follows, in paper covers:

Vols. I. to IV. (inclusive), each,	$3.00
Vols. V. to VIII. (inclusive), each,	4.00
Vols. IX. and X. (a small supply on hand, which will be sold only with complete sets, at a price of $10.00 each.)	
Vols. XI. to XXIX. (inclusive), each,	5.00
Vols. XXX. and XXXI., each,	6.00
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Complete set of *Transactions*, Vols. I. to XXXVIII., inclusive, half-morocco binding (freight prepaid),	234.00

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Per annum,	10.00
(To members of the Institute, public libraries, educational institutions and technical societies, $5.00.)	
Single numbers,	2.00
(To members of the Institute, public libraries, etc., $1.00.)	

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Indexes.

Index, Vols. XVI. to XX. (inclusive), paper, . . . $1.00
Index, Vols. XXI. to XXV. (inclusive), cloth, . . . 1.25
Index, Vols. XXVI. to XXX. (inclusive), cloth, $1.50, half-morocco, ... 2.50

Index, Vols. I. to XXXV. (inclusive).—This volume, an octavo of 706 pages, affords a ready and complete reference to any subject treated or alluded to in the *Transactions*, Vols. I. to XXXV., inclusive. The names of persons, mines, works, towns, etc., have been included; and abundant cross-references and classified sub-headings have been added to facilitate rapid consultation.

The Institute maintains at more than a hundred important mining centers throughout the world, free sets of its *Transactions*, open for consultation without fee, to all suitable applicants. Hence, the value of this index is by no means limited to individual possessors of complete sets of the *Transactions*. Moreover, the title of a paper, or the record of any remarks concerning a subject, being found in the Index, the Secretary's office of the Institute will supply upon written application any desired information as to the nature and length of said paper, whether it can be supplied in separate pamphlet form, etc.

Bound in cloth, $5.00, half-morocco, ... $6.00

Special Editions.

"*The Genesis of Ore-Deposits*," comprising the famous treatise of the late Professor Franz Posepny, with the successive discussions thereof by Le Conte, Blake, Winchell, Church, Emmons, Becker, Cazin, Rickard and Raymond (all of which were published in Volumes XXIII. and XXIV. of the *Transactions* of the Institute, and subsequently in the special "Posepny Volume," issued by the Institute); also, later papers by Van Hise, Emmons, Weed, Lindgren, Vogt, Kemp, Blake, Rickard and others, and the discussions of these papers by De Launay, Beck, and many others (some of these were included in Volume XXX. and the remainder appeared in Volume XXXI.); also a complete bibliography of Institute papers and discussions on this subject from 1871 to the year 1902.

The original Posepny volume comprised 265 pages, and was sold for $2.50, at which price the edition was long since exhausted. The present volume is an octavo of 825 pages.

Bound in cloth, $6.00, half-morocco, ... $7.00

"*The Evolution of Mine-Surveying Instruments.*" This is a volume of about 400 pages, containing the original paper of Mr. Dunbar D. Scott on that subject (*Transactions*, XXVIII.), first published in

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1898, together with later papers, continuing the same subject, and discussions thereof, by Hoskold, Lyman, Davis and many others.

Bound in cloth, $3.50, half-morocco, ... $4.50

Glossary of Mining and Metallurgical Terms (1881), cloth, . 1.00

Spanish-American Mining and Metallurgical Glossary, bound in leather, pocket-size, 96 pages, ... 0.75

List of Members, Rules, etc., paper, ... 0.50

Pamphlets.

1. The Minutes of the Proceedings of each Meeting.

2. Such of the papers presented or read by title at each Meeting as are furnished by the authors and approved by the Council for full publication. (In nearly all cases in which papers, the titles of which appear in the Proceedings, are not subsequently published, they have been withdrawn by the authors.) These papers are published separately in pamphlet form, and are marked "subject to revision." Beyond the edition distributed, without charge, to members and associates not in arrears, a small supply is retained to meet subsequent demand. There are no copies on hand of papers read before 1880. The stock is nearly complete from 1880. These papers are for sale at the office of the Secretary, or are sent to purchasers, charges paid, on receipt of the price, as follows:

No. of Pages.	Single Copies.	10 Copies.	20 Copies.
24 or less	$0.25	$2.00	$3.50
25 to 48	0.30	2.50	4.50
49 to 80	0.40	3.25	5.25
81 to 96	0.45	3.50	6.00
97 to 128	0.50	3.75	6.25
129 to 144	0.55	4.00	6.50
145 to 160	0.60	4.25	6.75
161 to 176	0.65	4.50	7.00

Papers with folders and inserted plates subject to special price.

Authors' Edition of Pamphlets.

Extra copies of pamphlets, if ordered before the printing of the *Bi-Monthly Bulletin*, will be furnished to members of the Institute at special rates, which will be given on application to the Assistant Secretary, Joseph Struthers, 29 West 39th St., New York, N. Y.

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CERTIFICATE OF INCORPORATION.

WE the undersigned, being all persons of full age and citizens of the United States, and a majority residents of the State of New York, desiring to form a corporation pursuant to the provisions of the Membership Corporations Law for the purpose of incorporating, as provided in Section 5 of Article I. of said law, the existing unincorporated association known as American Institute of Mining Engineers, do hereby make, acknowledge and file this Certificate for that purpose, and do Certify as follows:

I. That the American Institute of Mining Engineers is an unincorporated association organized and existing with the object of promoting the arts and sciences connected with the economic production of the useful minerals and metals and the welfare of those employed in these industries by means of meetings for social intercourse and the reading and discussion of professional papers, and to circulate by means of publications among its members and associates the information thus obtained.

II. That the persons duly appointed or designated to manage the affairs of said association are designated by the rules thereof Members of its Council; that the undersigned are all members of said Council as the same was constituted on the 29th day of December, 1904.

III. That on said last-mentioned date a regularly called meeting of said association was held at its office in the Borough of Manhattan, City of New York; that thirty days before such meeting notice of the intention to incorporate said association was given by mail to each member thereof whose residence or post-office address is known; that at said meeting the following resolutions were offered, seconded and duly adopted by the unanimous vote of all its members then present, to wit:

"*Resolved*, That it is the sense of the members and associates of the American Institute of Mining Engineers in general meeting assembled that it is desirable and necessary for the well being of said association and its members and for the furtherance of the objects for which the same has been formed, that said association incorporate under the Membership Corporations Law of the State of New York;

And Further Resolved, That the Members of the Council of this Association, or a majority thereof, be and they hereby are authorized, in accordance with the provisions of Section 5 of Article I. of the Membership Corporations Law, to incorporate this association for the same purposes for which it has been organized and conducted, in the manner provided in Article II. of said law;

And Further Resolved, That the name of said corporation as hereby adopted by this meeting shall be American Institute of Mining Engineers.

And Further Resolved, That the said incorporators shall be named in the Certificate of Incorporation as directors of such corporation until its first annual meeting, and that such directors and their successors in office shall be and they hereby are authorized to enact and adopt a Constitution and By-Laws for the government of said corporation."

IV. That the name of the proposed corporation is American Institute of Mining Engineers.

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V. That the purposes for which this corporation is to be formed are: To promote the arts and sciences connected with the economic production of the useful minerals and metals and the welfare of those employed in these industries by means of meetings for social intercourse, and the reading and discussion of professional papers, and to circulate by means of publications among its members the information thus obtained; and to establish and maintain a place of meeting for its members and a hall for the reading of papers and delivery of addresses, and a library of books relating to subjects cognate to the sciences and arts of mining and metallurgy.

VI. That the territory in which the operations of this corporation are to be principally conducted is the United States of America, and the principal office for the transaction of its affairs is to be located in the City, County and State of New York.

VII. That the duration of this corporation is to be perpetual.

VIII. That the number of directors of this corporation is to be nine. At its first annual meeting three directors shall be elected to serve for one year; three directors shall be elected to serve for two years, and three directors shall be elected to serve for three years. At each subsequent annual meeting so many directors only shall be elected as shall be necessary to fill vacancies then existing in the Board; and each director then chosen shall be elected to serve for three years, unless the vacancy which he shall be chosen to fill was caused otherwise than by expiration of the former incumbent's term of office, in which event he shall be elected to serve for the balance of his predecessor's unexpired term.

IX. That the time for holding the annual meeting of this corporation shall be the third Tuesday of February in each year.

X. That the names and post-office addresses of the subscribers hereto who are to be directors of this corporation until the first annual meeting are

Names.	Post-Office Addresses.
James Gayley, ...	71 Broadway, New York City;
Frank Lyman, ...	88 Wall Street, New York City;
James F. Kemp, ...	Columbia University, New York City;
Charles H. Snow, ...	New York University, New York City;
Frank Klepetko, ...	24 State Street, New York City;
Thomas A. Rickard, . . .	261 Broadway, New York City;
James Douglas, ...	99 John Street, New York City;
Albert R. Ledoux, ...	99 John Street, New York City;
Rossiter W. Raymond, . .	99 John Street, New York City.

The above certificate, dated December 30, 1904, was signed and acknowledged by the subscribers before William A. Lockwood, Notary Public (No. 74), New York County.

I, the undersigned Justice of the Supreme Court of the State of New York, do hereby approve of the above Certificate of Incorporation of the American Institute of Mining Engineers, and do authorize the same to be filed.

SAMUEL GREENBAUM,
Justice of the Supreme Court.

January 7, 1905.

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CONSTITUTION.

[ADOPTED FEB. 21, 1905.]

ARTICLE I.

NAME AND OBJECT.

SEC. 1. This Institute is incorporated under the Membership Corporation Law of the State of New York; its corporate name is AMERICAN INSTITUTE OF MINING ENGINEERS; and its objects are such as are stated in its Certificate of Incorporation.

ARTICLE II.

MEMBERS.

SEC. 1. The membership of the Institute shall comprise four classes, namely: (1) Members; (2) Honorary Members; (3) Associates; and (4) Honorary Associates. Only Members and Associates residing within the United States of America, Republic of Mexico and Dominion of Canada shall be entitled to vote at the meetings of the Institute.

SEC. 2. All Members, Honorary Members, Associates and Honorary Associates of the American Institute of Mining Engineers as the same existed on the day of the incorporation of this Institute, are Members, Honorary Members, Associates and Honorary Associates, respectively, of this Corporation.

SEC. 3. The following classes of persons shall be eligible for membership in the Institute, namely: as Members and Honorary Members, all professional mining engineers, geologists, metallurgists or chemists, and all persons practically engaged in mining, metallurgy or metallurgical engineering; as Associates and Honorary Associates, all persons desirous of being connected with the Institute who, in the opinion of the Council, are suitable.

SEC. 4. Every candidate for election as a Member or Associate of the Institute must be proposed for election by at least three Members or Associates; must be approved by the Committee on Membership, as prescribed in the By-Laws; and must be elected by the Council. Not less than three-fourths of the votes cast shall be necessary to an election. Every person so elected shall become a Member or Associate, as the case may be, upon payment of his first dues as hereinafter prescribed. Each candidate for Honorary Member or Honorary Associate, must be recommended by at least ten Members or Associates; must be approved by the Council; and must be elected by ballot at a meeting of the Board of Directors by the unanimous vote of all the Directors present; provided, however, that the number of Honorary Members and Honorary Associates shall not at any time exceed twenty.

SEC. 5. If any person elected a Member or Associate does not, within sixty days after notice of his election, accept the same and pay his initiation fee and dues for the current year, his election may be cancelled at the discretion of the Council.

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SEC. 6. The Council may at any time change the classification of a person elected as an Associate so as to make him a Member, or vice versa. All Members and Associates shall be equally entitled to the privileges of membership, provided that Honorary Members, Honorary Associates, and Members and Associates whose Post-Office addresses shall be outside of the United States, Mexico and Canada, shall not be entitled to vote.

ARTICLE III.

DUES.

SEC. 1. The dues of Members and Associates shall be Ten Dollars per annum, payable in advance on the first day of each Calendar year. Each newly elected Member or Associate shall pay, when notified of election, an initiation fee of Ten Dollars in addition to the dues for the current year. Honorary Members and Honorary Associates shall not be liable to initiation fee or dues. Any Member or Associate in arrears for one year may, at the discretion of the Council, be deprived of the receipt of publications or stricken from the list of Members, provided that he may be restored to membership by the Council on payment of all arrears or may be again proposed and elected after an interval of three years.

SEC. 2. Any Member or Associate not in arrears may become, by the payment of One Hundred and Fifty Dollars at one time, a Life Member or Associate; and shall not be liable thereafter to annual dues.

ARTICLE IV.

BUSINESS MEETINGS OF THE INSTITUTE.

SEC. 1. The annual meeting of the Institute for the election of Directors and transaction of other business shall take place on the third Tuesday in February in each year. A report of the financial condition of the Institute and an abstract of the accounts shall be furnished by the Directors, and presented at each annual meeting.

SEC. 2. Special business meetings of the Institute may be held at such times and places as the Board of Directors may appoint, upon notice to all Members and Associates entitled to vote, directed to each at his last known Post-Office address, and mailed in the City of New York not less than twenty days before the date fixed for such meeting.

SEC. 3. At all business meetings of the Institute the presence of nine Members and Associates shall constitute a quorum.

SEC. 4. At all business meetings of the Institute Members and Associates may vote either in person or by proxy, but no Member or Associate in arrears since the last annual meeting shall be entitled to vote.

ARTICLE V.

OTHER MEETINGS OF THE INSTITUTE.

SEC. 1. All meetings of the Institute other than business meetings shall be held at such times and places as the Council may appoint. Notice of all such meetings shall be given to all Members and Associates by mail.

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ARTICLE VI.

Directors and Officers.

Sec. 1. The business and financial affairs of the Institute shall be managed by a Board of Directors, who shall be elected at the annual meeting in the manner prescribed in the Certificate of Incorporation.

Sec. 2. The officers of the corporation shall be a President, Vice-President, Secretary and Treasurer, who shall be elected by the Directors from among their number. All such officers shall be elected at the first meeting of the Board of Directors after each annual meeting of the corporation, and shall hold office for one year or until their successors are elected and qualify.

The duties of all officers shall be such as usually pertain to their offices, respectively, together with such other duties as may from time to time be prescribed for them by the By-Laws. The Treasurer shall give a bond for the faithful performance of his duties in a sum to be fixed by the Board of Directors, but at the expense of the Institute.

Sec. 3. In the event of a vacancy occurring in the Board of Directors by death, resignation or otherwise, the remaining members of the Board may, by a majority vote, elect a successor to fill the vacancy, who shall continue in office until the next annual meeting or until his successor shall have been chosen.

Sec. 4. The Board of Directors may, in its discretion, declare the place of any Director vacant, on his failure for any reason, to attend three successive meetings of the Board. Any Director who shall under this section or in any other manner cease to be a member of the Board shall, at the same time, be held to have vacated any other office to which he shall previously have been elected; and the Board shall elect a new incumbent to the said vacant office.

Sec. 5. The Board of Directors may from time to time appoint from their own number standing and special committees, and may delegate to such committees such duties as they may see fit.

ARTICLE VII.

Meetings of the Board of Directors.

Sec. 1. A regular meeting of the Board of Directors for the election of officers and the transaction of other business shall be held on the third Tuesday in February in each year, after the adjournment of the annual meeting of the Institute.

Sec. 2. Special meetings of the Board of Directors, at which any business may be transacted, may be called to meet at any time at the office of the Institute in the City of New York, by notice in writing mailed at least five days before the meeting, by the Secretary to each member of the Board at his last known Post-Office address, signed either by the President or the Vice-President or by three members of the Board.

Sec. 3. At all meetings of the Board of Directors the presence of five members shall constitute a quorum.

ARTICLE VIII.

The Council.

Sec. 1. The professional, technical, scientific and social interests of the Institute shall be committed to the supervision of a Council composed of a President

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of the Council, six Vice-Presidents of the Council, a Secretary of the Council and nine Councilors, who shall be elected from among the Members and Associates of the Institute in the manner hereinafter prescribed. Members of the Council may or may not be members of the Board of Directors.

SEC. 2. The President of the Council shall be elected for one year, and no person shall be eligible for immediate re-election to this office who shall have held the same for two consecutive years.

After the first year Vice-Presidents of the Council shall be elected to serve for two years, and Councilors shall be elected to serve for three years. No Vice-President of the Council or Councilor shall be eligible for immediate re-election to the same office at the expiration of the term for which he was elected. The Secretary of the Council shall be elected annually.

SEC. 3. At the first annual meeting, to be held in the year 1905, there shall be elected a President of the Council to serve for one year, a Secretary of the Council to serve for one year, three Vice-Presidents of the Council to serve for one year, three Vice-Presidents of the Council to serve for two years, three Councilors to serve for one year, three Councilors to serve for two years, and three Councilors to serve for three years. At each subsequent annual meeting there shall be elected a President of the Council to serve for one year; a Secretary of the Council to serve for one year; three Vice-Presidents of the Council to serve for two years; and three Councilors to serve for three years. The term of office of all Members of the Council shall continue until the adjournment of the meeting at which their successors are elected.

SEC. 4. Vacancies in the Council may occur by death or resignation; or the Council may, by the vote of a majority of all its members, declare the place of any officer or member of the Council vacant, on his failure for one year, from inability or otherwise, to attend the regular meetings or perform the duties of his office. All vacancies shall be filled by the appointment of the Council, and any person so appointed shall hold office for the remainder of the term for which his predecessor was elected or appointed; *provided* that the said appointment shall not render such person ineligible for election to the Council at the next meeting.

SEC. 5. The presence of five members of the Council shall constitute a quorum; but the Council may appoint an Executive Committee, or any business coming within the authority of the Council may be transacted at a regularly-called meeting thereof, at which less than a quorum may be present, subject to the approval of a majority of the Council subsequently given in writing to the Secretary and recorded by him with the minutes.

SEC. 6. The election of the Council shall take place at the regular annual meeting of the Institute. Nominations for members of the Council may be sent in writing to the Secretary accompanied with the names of the proposers at any time not less than thirty days before the annual meeting; and the Secretary shall, not less than two weeks before said meeting, mail to every Member or Associate entitled to vote a list of all nominations for each office so received, together with the names of the persons ineligible for election to each office; and if the Council or a Committee thereof, appointed for the purpose, shall have recommended any nomination, such recommendation may also be sent to the Members and Associates with the list of all nominations made.

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ARTICLE IX.

Meetings of the Council.

Sec. 1. Meetings of the Council shall be held at such times and places as the President of the Council or one of the Vice-Presidents of the Council may appoint.

Sec. 2. A meeting of the Council may be held on the day of the annual meeting of the Institute without previous notice. Written notice of all other meetings of the Council, specifying the time and place of such meeting, signed by the Secretary, shall be mailed to every member of the Council at his last known Post-Office address at least ten days before the date of the meeting.

ARTICLE X.

Papers and Publications.

Sec. 1. The Council shall have power to decide as to the acceptance and publication of any professional papers presented to the Institute, subject to such conditions as the Board of Directors may prescribe.

Sec. 2. The copyright of all professional papers communicated to and accepted by the Institute shall be vested in it, unless otherwise expressly agreed between the Council and the author. The Institute shall not assume responsibility for any statements of fact or opinion advanced in the papers or discussions at its meetings. Neither the Council nor the Institute shall officially approve or disapprove any technical or scientific opinion or any proposed enterprise, outside of the management of the meetings, discussions and publications of the Institute, and the conduct of its business affairs by the Board of Directors.

Sec. 3. Special Committees may from time to time be appointed by the Council to make investigations and prepare reports for presentation to the Institute, but no action shall be taken binding the Institute for or against the conclusions embodied in any such reports.

ARTICLE XI.

Suspensions and Expulsions.

Sec. 1. Any member of the Institute who shall be convicted of a crime involving, in the opinion of the Board of Directors, moral turpitude, shall, upon the passage by the Board of Directors of a resolution declaring the crime for which he has been convicted to be of such character, be thereupon dropped from membership in this Institute.

Sec. 2. Any member of the Institute may be suspended or expelled for misconduct by the Board of Directors, after charges setting forth such misconduct shall have been prepared by the Council and filed in writing with the Board. Upon the receipt of such charges in writing, the Board may, in its discretion, suspend such member pending a hearing and determination thereupon. As soon as may be after the receipt of such charges, the Board shall fix a date for a hearing thereupon and shall give to the accused member notice thereof in writing, mailed to him at his last known Post-Office address not less than thirty days before said date, accompanied by a full copy of the charges and a copy of the second, third and fourth sections of this article.

Sec. 3. Upon the day fixed for the hearing, the accused member may appear before the Board, either in person or by an accredited representative; hear any

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witnesses who may be called in support of the charges and at his option cross-examine the same; and hear read any documentary evidence offered in support of the charges. The accused may, in his discretion, produce and examine witnesses in his defence, and submit documentary evidence, including a statement from himself in writing. After the conclusion of the hearing, the Board of Directors shall consider and vote to approve or disapprove the charges. If the Board shall, by a vote of two-thirds of its members, declare the charges sustained, it may suspend the member for a stated period or expel him.

SEC. 4. If the accused member shall not appear at the hearing, and shall within three months thereafter file with the Board an affidavit stating that he had not received notice of the charges against him in time to enable him to present his defence, the Board shall fix a date for a re-hearing within three months from the receipt of such affidavit and shall immediately notify the accused member by mail of such date. Upon the re-hearing, the accused shall have the same privilege of presenting his defence as he would have had upon the original hearing; and after the defence is presented, the Board shall take a new vote upon the charges, the result of which shall be conclusive.

SEC. 5. All interests in the property of the Institute of persons resigning, or otherwise ceasing to be Members or Associates, shall vest in the Institute.

ARTICLE XII.

AMENDMENTS.

SEC. 1. This Constitution or any Article or Section thereof may be amended at any annual meeting by a two-thirds vote of all the members present in person or by proxy, *provided* that notice of the proposed amendment shall have been given in writing at a previous meeting, and *provided also* that the amendment or amendments so adopted shall have been printed and mailed to all Members and Associates not later than thirty days before the annual meeting. Any amendment or amendments approved by a majority of the votes cast shall be deemed to have been adopted, and shall become a part of this Constitution. The Secretary shall forthwith print and distribute to Members and Associates an announcement of the result of said vote, and if any amendment or amendments shall have been adopted, a copy of the section or sections so amended.

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BY-LAWS.

[Adopted Feb. 21, 1905. Amended Feb. 20, 1906, and Nov. 16, 1906.]

I. Presiding Officers.

At all Business meetings of the Institute the President, or, in his absence, the Vice-President, or, in the absence of both of them, any other member of the Board of Directors to be chosen by the meeting, shall preside.

At all other meetings of the Institute the President of the Council or, in his absence, one of the Vice-Presidents, if present, shall preside.

II. Order of Business.

At each Business meeting of the Institute the order of business shall be as follows:

1. Reading of minutes of preceding meeting.
2. Report of the President.
3. Report of the Treasurer.
4. Report of the Secretary.
5. Election of Directors.
6. Election of Members of the Council.
7. Reports of Standing Committees.
8. Reports of Special Committees.
9. Special Orders.
10. Miscellaneous business.

This order of business may be changed by a vote of a majority of the Members and Associates present in person or by proxy.

The usual parliamentary rules shall govern all meetings of the Institute except in cases otherwise provided by the Constitution or the By-Laws.

At all sessions of the Institute other than business meetings, the order of proceedings and the time of adjournment shall rest in the discretion of the presiding officer.

III. Secretary.

The Secretary shall keep a record of the proceedings of all meetings of the Institute. He shall be custodian of the Corporate Seal, of the Minute Books, and of all Legal Documents belonging to the Institute. He shall conduct, on behalf of the Institute, all correspondence relating to business matters, except such as pertains directly to the office of the Treasurer.

He shall notify all officers and Directors and Members of the Council, and all Members of Committees of their election and appointment; shall issue notices of all meetings of the Board, and of the annual and other meetings of the Institute; and shall, in calling special meetings of the Directors, specify the object of such meeting.

IV. Secretary of the Council.

The Secretary of the Council shall act as the Clerk of that body at all of its meetings and at all meetings of the Institute called for the discussion of professional, technical or scientific matters, or for any other purpose than the transaction of business.

He shall be custodian of all technical or scientific papers submitted to the In-

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stitute for its consideration, shall have charge of the editing and printing of all material published by the Institute, and of the distribution thereof. On the first day of May following the year in which each volume of *Transactions* is printed, he shall turn over to the Library Committee all copies of the same not theretofore distributed by him. He shall have charge of all the correspondence of the Institute relating to other than business affairs.

The Secretary of the Council shall receive a salary to be fixed by the Board of Directors. He may appoint an Assistant with the title of Editor, who shall likewise receive a salary to be fixed by the Board of Directors.

The Secretary of the Council may or may not be the same person as the Secretary of the Institute.

V. Assistant Secretary.

The Secretary may, with the approval of the Board of Directors, appoint an Assistant to whom both he and the Secretary of the Council may delegate such of his or their duties as he or they may see fit. This Assistant Secretary shall receive such salary as shall be fixed by the Board of Directors, which shall cover his services both to the Secretary and to the Secretary of the Council.

VI. Treasurer.

The Treasurer shall collect and, under the direction of the Board of Directors, shall disburse all funds of the Institute. He shall keep regular accounts in books belonging to the Institute, which shall be open to any member of the Board of Directors. He shall report in writing at each annual meeting of the Institute and at every meeting of the Board of Directors at which such report shall be called for, the balance of money on hand, and any existing appropriation which may affect the same.

His accounts shall be audited annually by a Committee of three Members or Associates to be appointed by the President at least thirty days prior to the annual meeting in each year, which Committee shall report thereon at such annual meeting.

The Treasurer may, at his discretion, place funds of the Institute, not at any time exceeding $5,000, in a special account in a Bank or Trust Company, subject to the draft of the Assistant Treasurer, and may delegate to the Assistant Treasurer the duty of paying, out of this account, the current expenses of the Institute.

The Treasurer shall be solely responsible to the Institute for all moneys received, whether the same are entrusted to the Assistant Treasurer or not.

VII. Assistant Treasurer.

The Treasurer may appoint, with the approval of the Board of Directors, an Assistant Treasurer, to whom he may delegate the duty of conducting the correspondence incidental to the office of Treasurer, of receiving and depositing in bank to the credit of the Institute all moneys received, and of paying, out of the special account upon which he may be authorized to draw, the necessary expenses of the Institute. The Treasurer may require of him a bond, running to the Treasurer personally, in an amount not exceeding $5,000, the expense of which shall be borne by the Institute.

The Assistant Treasurer shall receive such compensation as shall be fixed by the Board of Directors.

The offices of the Assistant Secretary and of the Assistant Treasurer may, if

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so desired by both the Secretary and the Treasurer and approved by the Board of Directors, be united in the same person, who shall then receive the salary of both offices.

The Assistant Treasurer may, with the approval of the Board of Directors, employ such persons as are necessary to constitute a clerical and office force for himself, the Assistant Secretary and the Secretary of the Council, at such salaries as shall be approved by the Board of Directors. He shall, if the offices of Assistant Secretary and Assistant Treasurer be united in the same person, be the immediate superior of all such employees, unless the Secretary of the Council or the Treasurer be present, in which event either of them shall be the superior of all employees, including their respective assistants.

VIII. Standing Committees.

The Standing Committees of the Institute shall be three in number, known respectively as the Finance Committee, the Library Committee and the Committee on Membership.

The Finance Committee and the Library Committee shall each consist of three members of the Board of Directors, and shall be appointed by the President at the first meeting of the Board, after the annual meeting in each year.

The Committee on Membership shall consist of five Members of the Council, and shall be appointed by the President of the Council, at the first meeting of the Council after the first annual meeting in each year.

IX. Finance Committee.

It shall be the duty of the Finance Committee to inquire into and examine the financial condition of the Institute, and to consider ways and means of increasing its revenues and of limiting its expenses. It shall report from time to time to the Board as often as it may deem expedient, and whenever it shall be directed so to do; and the Treasurer shall at all times furnish it with such statements and information as it may desire.

It shall determine the investment of such surplus moneys as shall from time to time accrue to the Institute. It shall, at least once in each year, examine the securities belonging to the Institute in the custody of the Treasurer, and report thereon to the Board.

It may, at any time, examine the books and vouchers of the Treasurer and Assistant Treasurer.

The Treasurer shall not be a member of the Finance Committee, but shall attend the meetings of the same if requested to do so.

X. Library Committee.

The Library Committee shall be the custodian of all books in the Institute Library and of additions thereto; also of all back numbers of the *Transactions* of the Institute. It shall, on the first day of May, of each year, receive from the Secretary of the Council, and receipt for same to him, all the volumes of *Transactions* for the preceding year, not then distributed by said Secretary.

It shall cause to be kept, under the direction of the Assistant Secretary, a catalogue of all books in the Library and an account in ledger form of all volumes of *Transactions* in its custody, in which shall be charged to it all volumes delivered to it, and in which shall be credited all volumes taken from its custody for sale or for any other purpose.

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The receipts from the sale of any volume of *Transactions* taken from the custody of the LIBRARY COMMITTEE shall be credited to the LIBRARY COMMITTEE on the books of the Treasurer, and devoted to the general purposes of the Institute.

XI. COMMITTEE ON MEMBERSHIP.

All nominations for Members or Associates of the Institute shall be submitted to and passed upon by the COMMITTEE ON MEMBERSHIP, who shall report thereon to the Council. It shall receive and consider all communications respecting candidates, and shall make diligent inquiry as to the character and qualifications of each one. Its proceedings shall be secret and confidential.

No member of the Committee shall propose any candidate.

XII. ELECTION OF MEMBERS.

After the COMMITTEE ON MEMBERSHIP shall have reported to the Council its conclusions as to the acceptability of each candidate, the Council shall vote upon the same.

Two negative votes of members of the Council present shall prevent the election of any candidate. No person shall be proposed for election to the Institute within one year after his name shall have been rejected by the Council.

XIII. UNITED ENGINEERING SOCIETY.

The Board of Directors shall, at its first meeting after the adoption of these By-Laws, designate three Members or Associates of this Institute to be representatives of this Institute upon the Board of Trustees of the UNITED ENGINEERING SOCIETY, making at the same time provision for the expiration of the terms of office of said representatives, as provided in the By-Laws of the said UNITED ENGINEERING SOCIETY.

At the last meeting of the Board of Directors prior to the first day of each January thereafter, the Board shall designate a Member or Associate of this Institute to be a representative of this Institute upon the Board of Trustees of the said UNITED ENGINEERING SOCIETY for a period of three years beginning at the next ensuing annual meeting of said Society.

At any time when a vacancy shall occur in the representation of this Institute in the Board of Trustees of said Society, by reason of the death, resignation or removal of any such representative therein, the Board of Directors of this Institute shall designate a Member or Associate to fill such unexpired term.

XIV. PUBLICATIONS.

The publications of the Institute shall include a periodical, called the *Bi-Monthly Bulletin* of the American Institute of Mining Engineers, which shall contain reports of proceedings, professional papers, notices, and other matter of interest to members. From the annual dues paid by each Member or Associate, five dollars shall be deducted and applied as a subscription to the *Bi-Monthly Bulletin* for the year covered by such payment.

XV. AMENDMENTS.

These By-Laws may at any time be altered or amended by a vote of two-thirds of the Board of Directors, or by the Members, at a business meeting of the Institute, in the same manner provided for amendments of the Constitution in Article XII. thereof.

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ANNUAL MEETING.

The election of officers by vote of the members and associates in person or by proxy at the Annual Meeting, Feb. 18, 1907, resulted as follows:

COUNCIL.

President of the Council.

John Hays Hammond, ... New York, N. Y.
(Term expires February, 1908.)

Vice-Presidents of the Council.

Samuel B. Christy, ... Berkeley, Cal.
John A. Church, ... New York, N. Y.
Persifor Frazer, ... Philadelphia, Pa.
(Term expires February, 1909.)

Councilors.

B. F. Fackenthal, Jr., ... Easton, Pa.
H. O. Hofman, ... Boston, Mass.
Walter R. Ingalls, ... New York, N. Y.
(Term expires February, 1910.)

Secretary of the Council.

R. W. Raymond, ... New York, N. Y.
(Term expires February, 1908.)

Assistant Secretary and Editor (by appointment).

Joseph Struthers, ... New York, N. Y.

DIRECTORS OF THE CORPORATION.

James Gayley, ... New York, N. Y.
Charles Kirchhoff, ... New York, N. Y.
Frank Lyman. . . . . New York, N. Y.
(Term expires February, 1910.)

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ACTS OF THE BOARD OF DIRECTORS.

At a meeting held May 10, 1906, the Secretary reported the death, on April 16, 1906, of Mr. H. W. B. Howard, Assistant Secretary and Assistant Treasurer of the Institute; also that by direction of the Council he had sent a copy of the resolution expressing the sympathy and regret of the Institute to the family of Mr. Howard.

At the same meeting the Secretary and the Treasurer reported respectively their appointment of Dr. Joseph Struthers as Assistant Secretary and Assistant Treasurer, to fill the vacancies created by the death of Mr. Howard, and the said appointments were confirmed.

At the same meeting, Mr. Robert A. Hadfield, Sheffield, England, and Mr. John E. Stead, Middlesbrough, England, having been recommended by unanimous vote of the Council, were unanimously elected Honorary Members, in recognition of their distinguished services to the arts and professions represented by the Institute.

At the same meeting the Secretary presented the case of members of the Institute, Public Libraries, etc., located in San Francisco and vicinity, and it was

Voted: That the Secretary be authorized in his discretion, at an average of one-half the usual price, to replace to members of the Institute, such publications thereof as have been destroyed by the late catastrophe in California, and, in his discretion, to replace by gift the complete sets of the *Transactions* formerly kept in libraries open to free public use, and by the State Mining Bureau of California and the California Miners' Association. And in any case of peculiar hardship or merit, the Secretary is authorized, with the approval of the President and the Treasurer, and of the Library Committee, to make further reduction of prices to members who have lost their copies of Institute publications as aforesaid.

At the meeting held Nov. 6, 1906, By-Law XIV. was amended by changing the number thereof to XV., and substituting therein the words "two-thirds" instead of "three-fourths," the By-Law, as amended, reading in full:

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XV. Amendments.

These By-Laws may at any time be altered or amended by a vote of two-thirds of the Board of Directors, or by the members, at a business meeting of the Institute, in the same manner provided for amendments of the Constitution in Article XII. thereof.

At the same meeting a new By-Law was added, as follows:

XIV. Publications.

The publications of the Institute shall include a periodical called the *Bi-Monthly Bulletin* of the American Institute of Mining Engineers, which shall contain reports of proceedings, professional papers, notices, and other matters of interest to members. From the annual dues paid by each member or associate, five dollars shall be deducted and applied as a subscription to the *Bi-Monthly Bulletin* for the year covered by such payment.

At the same meeting, the action of the Council providing for the issue and the circulation of the *Bi-Monthly Bulletin* and the procurement and insertion of suitable advertisements therein, was approved.

At a meeting held Feb. 19, 1907, the following officers were elected for the ensuing year: *President*, James Gayley; *Vice-President*, James Douglas; *Secretary*, R. W. Raymond; *Treasurer*, Frank Lyman.

Financial Statement.

The following statement of receipts and disbursements from Jan. 1 to Dec. 31, 1906, is published by authority of the Board of Directors:

Receipts.

Balance from statement of January, 1906,		$6,818.96
Annual dues,	$36,324.36	
Life memberships,	3,440.00	
Initiation fees,	2,960.20	
Binding of *Transactions*,	3,493.12	
Sale of publications,	3,000.62	
Electrotypes,	42.00	
Miscellaneous receipts,	166.97	
Advertising,	15.00	
		49,442.27
Carried forward,		$56,261.23

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Brought forward,		$56,261.23
Interest on bonds and deposits,		1,196.45
Reimbursement from Special Fund for installments paid to the United Engineering Society,		43,000.00
Reimbursement from Library Fund for library additions (for 1905 and 1906),		1,041.67
		$101,499.35

DISBURSEMENTS.

Printing Vol. XXXVI. of the *Transactions*, *Bi-Monthly Bulletin* and extra pamphlets,	$9,619.71	
Printing circulars and ballots,	230.25	
Binding Vol. XXXVI. of the *Transactions*,	2,925.00	
Binding miscellaneous volumes,	450.75	
Binding of exchanges,	357.20	
Engraving and electrotyping,	1,098.05	
Secretary's department, including clerks, stenographers and expenses of editing and proof-reading,	8,664.66	
Treasurer's department, including collection of dues, shipping, etc.,	7,126.25	
Librarian and assistant,	1,340.00	
Postage,	1,915.11	
Stationery,	766.07	
Rent,	2,500.00	
Express and freight charges,	1,212.67	
Telephone,	248.55	
Telegrams, cables, carfares,	65.98	
Office supplies and repairs,	145.62	
Storage of *Transactions*,	187.98	
Refunding over-payments,	16.00	
Insurance premiums,	271.43	
Collection charges,	35.47	
Extra clerical assistance,	132.50	
Special stenographers and expense of meetings, etc.,	1,131.92	
Auditing,	125.00	
Advertising expenses,	19.72	
Office cleaning and sundry expenses,	135.75	
		$40,721.64
Interest, being proportion due for 1905 and 1906 on land mortgage loan to Andrew Carnegie on Engineering Society's property,		$9,133.33
Office equipment,		41.95
Library additions,		363.65
United Engineering Society, instalment payments under agreement,		50,000.00
Balance,		1,238.78
		$101,499.35

We have examined the above statement, compared it with the books and vouchers and find same correct.

(Signed) BARROW, WADE, GUTHRIE & CO.,
Certified Public Accountants.

NEW YORK CITY, January 30, 1907.

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REPORT OF THE COUNCIL FOR THE YEAR 1906.

Meetings.

Two meetings for the reading and discussion of papers, etc., have been held during the year 1906—namely, the Ninetieth meeting, held February 21 to 24 at Bethlehem, Pa., and the Ninety-first meeting, held July 24 to 27 in London, England, in connection with the meeting of the Iron and Steel Institute.

The proceedings of these meetings, including descriptions of the entertainments and excursions connected therewith, have already been published and distributed to the members of the Institute; the Bethlehem meeting in *Bi-Monthly Bulletin*, No. 9, May, 1906, pp. 497 to 508, and the London meeting in *Bi-Monthly Bulletin*, No. 12, November, 1906, pp. 809 to 908. The proceedings of the London meeting were published separately and distributed among the many individuals and concerns whose cordial friendship and delightful hospitality contributed so largely to the success of the visits in London and environs and the excursions in England, Scotland and Germany.

These meetings were highly successful both professionally, on account of the large number of valuable papers presented, and socially through the various delightful excursions and entertainments provided for the visiting members and guests.

Publications.

Transactions.—Volume XXXVI. of the *Transactions*, an octavo of 1,001 pages, was issued and distributed in June, 1906.

Bi-Monthly Bulletin.—Six numbers of the *Bi-Monthly Bulletin* (Nos. 7 to 12), containing 1164 pages of technical papers and 140 pages of announcements (total, 1204 pages), were published and distributed promptly throughout the year 1906.

Membership.

Changes in membership have taken place during the year as follows:—283 members and 13 associates have been elected;

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3 members have been elected honorary members, and 17 associates have become members; the deaths of 40 members and 2 associates have been reported; 29 members and 3 associates have resigned; and 56 members and 6 associates have been dropped from the roll by reason of non-payment of dues, loss of correct address, etc.* These changes are shown in the accompanying table.

The total membership on Jan. 1, 1907, was 4,048, as compared with 3,884 on Jan. 1, 1906—a net gain for the year of 164 members.

Membership of the American Institute of Mining Engineers, Jan. 1, 1907.

	Honorary Members.	Members.	Associates.	Totals.
Membership Dec. 31, 1905	8	3,682	194	3,884
Gains: By Election	.....	283	13	296
Change of Status	3	17	.....	20
Reinstatement	.....	3	.....	3
Re-election	.....	1	.....	1
Losses: By Resignation	.....	29	3	32
Change of Status	.....	3	17	20
Dropping	.....	56	6	62
Death	.....	40	2	42
Total gains	3	304	13	320
Total losses	.....	128	28	156
Membership Dec. 31, 1906	11	3,858	179	4,048

The list of deaths reported during the year 1906 comprises the following names, the figures in parentheses indicating the year in which the persons named were elected to membership:

Members and Associates.—Arthur Vaughan Abbott (1882), William Anderson Akers (1889), R. Scott Allen (1905), George H. Arlett (1900), Thomas Septimus Austin (1883), William Tittley Batchelor (1902), Charles Lothian Bell (1897), Edgar Vallentine Bensusan (1892), Ernst Elmer Breisch (1896), Horace F. Brown (1895), James A. Burden (1876), Alexander B. Coxe (1880), George A. Crocker (1879), John Herbert Fraser (1904), Robert Gibson (1892), R. G. Hart (1900), Alexander W. Jolly (1899), George L. Keener (1899), John G. Lanning (1893), Nicholas Lennig (1882), Frank C. Mandell

* Many of these, no doubt, will be reinstated, as has been the case in former years.

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(1905), Edmund H. Miller (1895), John Fossbrook Morris (1903), F. J. Odling (1893), William Orr (1897), William Painter (1893), Stanley H. Pearce (1896), Herman Poole (1900), Ricardo G. Ramos (1898), Arthur F. Rising (1904), George H. Robinson (1886), Albert W. Sayles (1905), Richard J. Seddon (1888), James C. Simpson (1887), Francis Lewis Sperry (1889), John Stanton (1877), Edward G. Stoiber (1877), Samuel Thomas (1871), Abel Hyde Toll (1900), John Price Wetherill (1896), Harvey Ladew Williams (1903), Lawrence F. J. Wrinkle (1901).

[NOTE.—A Biographical Notice of Alexander B. Coxe was published in Vol. XXXVII. of the *Transactions*, and a similar notice of Thomas S. Austin appears elsewhere in the present volume. Concerning the remaining names in the above list, such data as the Secretary was able to obtain appeared in *Bi-Monthly Bulletin*, No. 16, July, 1907, under the title, "Biographical Notices of 1906."—J. S.]

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MEMBERSHIP.

The following list comprises the names of those persons elected as members, who duly accepted election during the year 1907. The marks used to designate the different classes of membership are: Life Member, **; Member, *; Associate Member, †.

†Max F. Abbé, ... New York, N. Y.
†Paul O. Abbé, ... New York, N. Y.
*Samuel J. Alderman, ... Benton, Cal.
*John A. Allen, ... Cleveland, Transvaal, So. Africa.
*Alexander Anderson, ... Edinburgh, Scotland.
*Lyndon K. Armstrong, ... Spokane, Wash.
*Charles E. Le N. Arnold, ... Ely, Nev.
*Raymond Auzias-Turenne, ... Seattle, Wash.
*George S. Backus, ... Smuggler, Colo.
*Rufus M. Bagg, Jr., ... Colorado Springs, Colo.
*Sydney H. Ball, ... Luebo, Congo Free State, W. Africa.
*Albert B. Bartlett, ... Cheyenne, Wyo.
†William S. Bartlett, ... London, England.
*Astolfo Bartoccini, ... Monterey, N. L., Mexico.
*Mowry Bates, ... New York, N. Y.
*James C. Bayles, ... New York, N. Y.
*Albert S. E. Beall, ... San Diego, Cal.
*Roger L. Beals, ... Rosario, Sinaloa, Mexico.
*Sir Hugh Bell, ... Northallerton, England.
*W. F. B. Berger, ... Leadville, Colo.
*Edwin S. Berry, ... San Francisco, Cal.
*John L. W. Birkinbine, ... Mexico City, Mexico.
*Harry H. Blackburn, ... Grottoes, Va.
*William H. Blackburn, ... Tonopah, Nev.
*William D. Blackmer, ... Rhyolite, Nev.
*Allen F. Blair, ... Tacoma, Wash.
*Herman Blumenau, ... Frankfort, Germany.
*J. F. Body, ... Philadelphia, Pa.
*Heinrich G. Boker, ... Remscheid, Germany.
*Francis C. Bowman, ... Smuggler, Colo.
*Reginald H. T. Brook, ... Ballarat, Victoria, Australia.
*Frank H. Brown, ... Salt Lake City, Utah.
*Harry L. Brown, ... Ames, Colo.
*John T. Brown, Jr., ... Pittsburg, Pa.
*Joseph Brown, ... Socorro, New Mexico.
*Walter S. Brown, ... Denver, Colo.
†William C. Bullitt, ... Philadelphia, Pa.

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*Jose Campa, ... Mexico City, Mexico.
*Donald F. Campbell, ... Baird P. O., Cal.
*Thomas Cantley, ... New Glasgow, N. S., Canada.
*Adolfo Cardenas, ... Punta Arenas, Costa Rica, Cent. Am.
*John R. Chamberlain, ... New York, N. Y.
*Frank M. Chambers, ... Columbia, Nev.
*Temple Chapman, ... Webb City, Mo.
*John A. Church, Jr., ... Guanajuato, Mexico.
*John L. Church, ... Cobalt, Ont., Canada.
*Frederick H. Clark, ... Guanajuato, Mexico.
**George A. Cleland, ... Tonopah, Nev.
*William T. Climo, ... Los Angeles, Cal.
*Lionel H. Cole, ... Rossland, B. C., Canada.
*Glenville A. Collins, ... Seattle, Wash.
*George M. Colvocoresses, ... New York, N. Y.
*Frederick S. Cook, ... Joplin, Mo.
*Charles E. Coote, ... Launceston, Tasmania.
*Edward T. Corkill, ... Toronto, Canada.
*Frederick Corkill, ... Tonopah, Nev.
*Fritz Cremer, ... South Chicago, Ill.
*Charles C. Crismon, ... Salt Lake City, Utah.
*Frederick G. Crist, ... San Dimas, Dur., Mexico.
†Cecil H. Cropper, ... Kalgoorlie, Western Australia.
*Raymond B. Crowell, ... Carson City, Nev.
*Timothy I. Crowley, ... Hornitos, Cal.
*J. Richmond Crum, ... Mexico City, Mexico.
*W. L. Cumings, ... South Bethlehem, Pa.
*Harle O. Cummins, ... Shasta, Cal.
*Floyd E. Cunnyngham, ... Pennington Gap, Va.
*Bracey Curtis, ... Nogales, Ariz.
*Frank M. Curtis, ... Los Angeles, Cal.
*Jack Cussons, ... Mineral, Va.
*Mark Daniel, ... Baker City, Ore.
*Percy W. Dargin, ... Rhyolite, Nev.
*Clifford G. Dennis, ... Ehrenberg, Ariz.
*E. H. De Vore, ... El Paso, Texas.
*Archibald A. C. Dickson, ... Debour, Rejoulie P. O., India.
*Emil E. Dieffenbach, ... Newark, N. J.
*Earl Dissinger, ... Canal Fulton, Ohio.
*Thomas F. Donnelly, ... El Oro, Mexico.
*Albert H. Donnewald, ... St. Louis, Mo.
*Francis V. Drake, ... Bishop, Cal.
*Robert I. Dunn, ... Seattle, Wash.
*Carl B. Dunster, ... Marquette, Mich.
*Elmer W. Durfee, ... Congress Junction, Ariz.
*William F. Dutton, ... New York, N. Y.
*George M. Dyott, ... Pittsburg, Pa.
*George R. D. Easley, ... Mackay, Idaho.
*Adrian D. Eatherly, ... Chilton, W. Va.
*Edwin E. Ellis, ... Galena, Ill.
*Howard L. Elton, ... Oaxaca, Oaxaca, Mexico.
*William H. Emmons, ... Washington, D. C.

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*Frederick Ewing, ... New Haven, Conn.
*Frederick G. Farish, ... Denver, Colo.
*Aubrey P. Fellows, ... Joplin, Mo.
*Norman R. Fisher, ... Haileybury, Ont., Canada.
*Edward F. Fitzhugh, ... Butte, Mont.
*Frederick B. Flinn, ... East Orange, N. J.
*Ernest E. Ford, ... Alhambra, Cal.
*Isaac P. Fraizer, ... Rolla, Mo.
*Albert Frank, ... Zacatecas, Mexico.
*Gay N. Freeman, ... Thermopolis, Wyo.
*Thomas Fudge, ... Bayonne, N. J.
*John H. Gallagher, ... Elko, Nev.
*Carroll D. Galvin, ... Elko, Nev.
*Fortuno P. Gamba, ... Pasto, Colombia, So. America.
*George T. Gambrill, Jr., ... Baltimore, Md.
*Russell T. Gard, ... New Haven, Conn.
*Herbert W. Gartrell, ... Bisbee, Ariz.
*John C. Gibson, ... Penon Blanco, Dur., Mexico.
*Charles E. Gilman, ... San Francisco, Cal.
*Kenneth L. Gilson, ... Trondhjem, Norway.
*Frank W. Giroux, ... Mayer, Ariz.
*Julius Goldberg, ... Great Falls, Mont.
*Lyman F. Gordon, ... Worcester, Mass.
*James H. Gray, ... New York, N. Y.
*George Griggs, ... Chihuahua, Mexico.
*Clyde T. Griswold, ... Colorado Springs, Colo.
*Thomas C. Groom, ... Cobar, N. S. W., Australia.
*Herbert C. Hale, ... Cleveland, Ohio.
*William H. Hale, ... Whitcomb, Mont.
*Carl P. Halter, ... Chihuahua, Mexico.
*John S. Hamman, ... Rhyolite, Nev.
*Robert S. Hanckel, ... San José, Costa Rica, Cent. Amer.
*Ernest P. Hargraves, ... Peak Hill, Western Australia.
*George B. Harrington, ... Elko, Nev.
*Edwin F. Harris, ... Tucson, Ariz.
*William S. Harrison, ... Parral, Mexico.
*Kiohey Hasegawa, ... Bisbee, Ariz.
†William M. Hauck, ... New York, N. Y.
*Carle R. Hayward, ... Quincy, Mass.
*Stanley Hayward, ... Prescott, Ariz.
*Paul H. Hebb, ... Tacoma, Wash.
*Carl A. Heberlein, ... Torreon, Coah., Mexico.
*Joseph H. Hedges, ... Guanacevi, Dur., Mexico.
*Hans L. Heldt, ... Zacatecas, Mexico.
*A. Roy Heise, ... Wadsworth, Nev.
*Enoch Henderson, ... Matchwood, Mich.
*Paul L. T. Heroult, ... New York, N. Y.
*Herbert T. Herr, ... Denver, Colo.
*Frank L. Hess, ... Washington, D. C.
*Hiram W. Hixon, ... Victoria Mine, Ont., Canada.
†Walter R. Hodge, ... Mexico City, Mexico.
*Geoffrey C. Hollis, ... Mertola, Portugal.

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*Preston K. Horner, ... Ely, Nev.
*J. Banks Hudson, ... Gadsden, Ala.
*William McM. Huff, ... Kellogg, Idaho.
*Wilson W. Hughes, ... Guanajuato, Mexico.
*James Humes, ... Basin, Mont.
*John M. Humphrey, ... Centralia, Pa.
*Charles H. Humphreys, ... Mt. Morgan, Queensland, Aust.
*Charles L. Huston, ... Coatesville, Pa.
*Henry M. Huxley, ... New York, N. Y.
†Lee Emmet Ives, ... Houghton, Mich.
*Paul Iweins, ... Paris, France.
*Cyril H. James, ... Melbourne, Victoria, Australia.
*Frank E. Johnesse, ... Lucile, Idaho.
*Dion L. Johnson, ... Duquesne, Ariz.
*J. Frank Johnston, ... Bethlehem, Pa.
*A. C. de Jongh, ... Nijmegen, Holland.
*Henry B. Kaeding, ... Cosala, Sinaloa, Mexico.
*Berthold Kapelowitz, ... Germiston, Transvaal, So. Africa.
*Ralph M. Kellogg, ... Searchlight, Nev.
*Spencer Kellogg, ... Buffalo, N. Y.
*Alfred Kimber, ... New York, N. Y.
*F. Foster Kip, ... Temosachic, Chih., Mexico.
*William A. Kissam, ... Manila, P. I.
*Cyril W. Knight, ... Toronto, Ont., Canada.
*Julius Kruttschnitt, Jr., ... Chicago, Ill.
*Wilbur G. Laird, ... New York, N. Y.
*Beach A. Laselle, ... Barkerville, B. C., Canada.
*Frederick G. Lasier, ... Birmingham, Mich.
*Harold N. Laurie, ... Perdue, Ore.
*James Lea, ... Johannesburg, Transvaal, So. Africa.
†John A. Leavell, ... Temple, Ariz.
*R. G. Edwards Leckie, ... Cobalt, Ont., Canada,
*Charles B. Lessner. . . . . Carrill, Spain.
*Robert S. Lewis, ... Morenci, Ariz.
*William E. L'Hame, ... Covelo, Cal.
*Carl O. Lindberg, ... Mexico City, Mexico.
*James Lindsay-Oliver, ... Chicago, Ill.
*William J. Linn, ... Chicago, Ill.
*William H. Linney, ... Cobalt, Ont., Canada.
*Hans v. und z. Loewenstein, ... Essen-Ruhr, Germany.
*Frank C. Loring, ... Toronto, Canada.
*Kenneth W. McComas, ... Perth Amboy, N. J.
*Hoyt S. McComb, ... Hondo, Coah., Mexico.
*James E. McDevitt, ... Youngstown, Ohio.
*Benjamin E. McKechnie, ... Lebanon, Pa.
*Walter F. McNeill, ... Alberta, Can.
*Charles T. Malcolmson, ... Norfolk, Va.
*John F. Manning, ... Chicksan, Korea.
*Stewart M. Marshall, ... Johnstown, Pa.
*John E. Masters, ... Silver City, Idaho.
*Ronald V. A. Mills, ... San Pedro, New Mexico.
*George Miltenberger, Jr., ... Philipsburg, Mont.

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*Maxwell C. Milton, ... Bisbee, Ariz.
*John A. Mitchell, ... Haileybury, Ont., Canada.
*William S. Mitchell, ... Haileybury, Ont., Canada.
*Robert McI. Moore, ... Kiandra, N. S. W., Australia.
*Roy Webb Moore, ... Tucson, Ariz.
*McHenry Mosier, ... Bisbee, Ariz.
*William D. B. Motter, Jr., ... Santa Barbara, Chih., Mexico.
**John M. Moubray, Sable Antelope Mine, N. W. Rhodesia, So. Africa.
*Stephen E. Murphy, ... Mancelona, Mich.
*William F. Murray, ... Denver, Colo.
*Kiutaro Nagai, ... Tokio, Japan.
*Keijiro Nakamura, ... Niigun, Iyo, Japan.
*Francisco Narvaez, ... Pachuca, Hidalgo, Mexico.
*D. W. C. Nelson, ... Baker City, Ore.
*John C. Nicholls, ... Chittabalbie, Korea.
*Edward G. Norton, ... Edinburg, Ind.
*Lewis Oliver, Jr., ... Newhouse, Utah.
*George A. Orrok, ... New York, N. Y.
*Vance C. Osmont, ... Dayton, Nev.
*William T. Oster, ... Custer, Idaho.
*Sidney W. Ough, ... Guanajuato, Mexico.
*Arthur R. Parsons, ... Tonopah, Nev.
*William C. W. Pearce, ... Irvinebank, No. Queensland, Aust.
*Francis J. Peck, ... Cleveland, Ohio.
*Frederick S. Pheby, ... Ely, Nev.
*Louis Phillip, ... Rio de Janeiro, Brazil, So. Amer.
*Frederick J. Platt, ... Scranton, Pa.
*Cecil W. Pocock, ... Bayonne, N. J.
*Howard A. Poillon, ... Snowden, Cal.
*Julio Posada, ... Chihuahua, Mexico.
*Edmund A. Prentis, Jr., ... New York, N. Y.
*Charles A. Pringle, ... San Isidro, Chih., Mexico.
*Henry T. Purdy, ... Punta Arenas, Costa Rica, Cent. Am.
*Lorin T. Putnam, ... Zeigler, Ill.
*John N. Reese, ... Harrisburg, Pa.
*Max Reichard, ... Saxony, Germany.
*Henry D. G. Reynolds, ... Blanchard, Ariz.
*James W. Rhew, ... Sinaloa, Mexico.
*Fred N. Rhodes, ... Bisbee, Ariz.
*Brent N. Rickard, ... Mapimi, Dur., Mexico.
†Glen A. Ricks, ... Houghton, Mich.
*Fred E. Righter, ... Hymera, Ind.
*Heaton R. Robertson, ... New Haven, Conn.
*John H. Robertson, ... Joplin, Mo.
*Max Roesler, ... Bisbee, Ariz.
*George E. Rose, ... Chicago, Ill.
*Alexander J. M. Ross, ... Blair, Nev.
*Frank A. Ross, ... Hedley, B. C., Canada.
*James G. Ross, ... Jersey City, N. J.
*David B. Rushmore, ... Schenectady, N. Y.
*William C. Russell, ... Tonopah, Nev.
*Edward F. Schaefer, ... New York, N. Y.

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*C. Quimby Schlereth, ... Durango, Mexico.
*Oswold N. Scott, ... Toronto, Ont., Canada.
†George Sealy, ... Galveston, Texas.
*Stanley C. Sears, ... Boston, Mass.
*John C. Sevier, ... Columbia, Nev.
*Frederic H. Sexton, ... Halifax, Nova Scotia, Canada.
*Rudolf von Seyfried, ... Newark, N. J.
*Millard K. Shaler, ... Lawrence, Kan.
*James B. Shaw, ... Mt. Morgan, Queensland, Aust.
*S. F. Shaw, ... Santa Barbara, Chih., Mexico.
*Waldo Sheldon, ... Greenwich, Conn.
*Alfred C. Sieboth, ... Florence, Ariz.
*Trevor B. Simon, ... Columbus, Ohio.
*Robert D. Skelley, ... Ishpeming, Mich.
*Henry B. Skewes, ... Grass Valley, Cal.
†Alexander H. Smith, ... New York, N. Y.
*Frank G. D. Smith, ... Batopilas, Chih., Mexico.
*Stewart K. Smith, ... Terre Haute, Ind.
*Frederick W. Snow, ... Great Falls, Mont.
**Tom R. Starkey, ... Punta Arenas, Chile, So. Amer.
*Charles J. Steffens, ... New York, N. Y.
*H. J. Stehli, ... New York, N. Y.
*Lawrence P. Stevens, ... Ironwood, Mich.
*Morton Stevens, ... Philadelphia, Pa.
*Paul X. Stoffel, ... Mapimi, Durango, Mexico.
*Edgar D. Stone, ... Etna, Ga.
*Leonard C. Stuckey, Dulcinea Mine, near Copaipo, Chile, So. America.
*William S. Sultan, ... Globe, Ariz.
*Henry N. Thomson, ... Anaconda, Mont.
*Robert W. Thomson, ... Toronto, Ont., Canada.
*B. Leonard Thorne, ... Hosmer, B. C., Canada.
*William H. Tolman, ... New York, N. Y.
*Neville F. Townsend, ... Rossland, B. C., Canada.
*Paul B. Tracy, ... Bingham Canyon, Utah.
*William E. Tracy, ... Telluride, Colo.
*Omar A. Turney, ... Phoenix, Ariz.
*Harold Turrell, ... Lomagurdi, Rhodesia, So. Africa.
*Henry Twynam, ... Cairns, N. Queensland, Australia.
*William J. Uren, ... Calumet, Mich.
*Francisco R. Varela, ... Zacatecas, Mexico.
*Anders B. Villadsen, ... Flat River, Mo.
*Edward Walker, ... London, E. C., England.
*Myron R. Walker, ... Ezutlan, Oax., Mexico.
*Robert Wallace, ... Eulalia, Chih., Mexico.
*Julius H. Warner, ... Haileybury, Ont., Can.
*Henry L. J. Warren, ... Salt Lake City, Utah.
*Henry S. Washington, ... Locust, N. J.
*Edward Watson, ... Akmolinsk, Siberia.
*Russell G. Wayland, ... Treadwell, Alaska.
*Clement L. Webster, ... Charles City, Iowa.
*Frederick R. Weeks, ... New York, N. Y.
*Alf. Welhaven, ... Chittabalbie, Korea.

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*Irving H. Wentworth,	Matehuala, S. L. P., Mexico.
*Wilfred F. Wheeler,	Urbana, Ill.
*Henry P. Wherry,	Indé, Durango, Mexico.
*Philip R. Whitman,	Concheno, Chih., Mexico.
*Francis G. Wickware,	New York, N. Y.
*Thomas B. Wilde,	Nogales, Ariz.
*Edwin P. Williams,	South Chicago, Ill.
*James M. Williams, Jr.,	Sierra Mojada, Coah., Mexico.
*George B. Wilson,	Searchlight, Nev.
*Clarence Woods,	Dewey, Idaho.
*Charles W. Workman,	Jalisco, Mexico.

Deaths.

The following list comprises the names of members whose deaths have been reported to the Secretary of the Institute during the year 1907:

Date of Election.	Name.	Date of Decease.
1887.	*Baker, Thomas T.,	April 30, 1906.
1897.	*Benton, Charles W.,	September 9, 1907.
1886.	*Bildt, Carl Wilhelm,	May 5, 1906.
1897.	*Blatchford, John,	August 10, 1906.
1888.	*Blue, John,	November 2, 1907.
1882.	*Boggs, William Robertson, Jr.,	November 17, 1907.
1874.	*Bradley, George L.,	March 26, 1906.
1886.	*Brown, Thomas Forster,	October 23, 1907.
1882.	*Burlingame, Eugene E.,	March 20, 1907.
1896.	*Butler, Chauncey E.,	January 26, 1907.
1903.	*Devinny, George V.,	August 22, 1907.
1898.	*Evans, George Henry,	February 4, 1907.
1887.	*Eveleth, J. K.,	December 6, 1907.
1886.	*Fiero, Albert W.,	July 28, 1906.
1893.	*Garrett, William Warren,	January 14, 1907.
1881.	*Glenn, William,	February 16, 1907.
1887.	*Hanna, George Byron,	May 21, 1906.
1877.	*Harrington, Bernard James,	November 29, 1907.
1874.	*Hearne, Frank J.,	February 25, 1907.
1900.	*Henne, Christopher,	December 12, 1906.
1883.	*Hewett, George C.,	August 12, 1907.
1888.	*Hillard, Charles J.,	September 12, 1907.
1898.	*Howard, Karl,	July 4, 1907.
1899.	†Hurley, Thomas J.,	December 13, 1907.
1894.	**Ingram, H. E.,	September 12, 1907.
1904.	*Johns, Thomas Edward,	November —, 1906.
1901.	†Johnston, William J.,	April 28, 1907.
1888.	*Jones, James F.,	November 7, 1907.
1872.	*Keyes, Winfield Scott,	December 12, 1906.
1892.	*Lehmann, Gustavus W.,	August —, 1906.

* Member. ** Life Member. † Associate.

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Date of Election.	Name.	Date of Decease.
1891.	*Leisenring, Walter,	January 5, 1907.
1882.	*Long, William H.,	October 27, 1906.
1904.	*Matthews, Frederick William,	——— —, 1906.
1900.	*Mercur, Robert Sayre,	April 22, 1907.
1887.	*Molson, Charles A.,	November 26, 1907.
1872.	*Neilson, William G.,	December 29, 1906.
1894.	*Pelatan, Louis,	June 8, 1907.
1904.	*Rock, Alfred Mayer,	August 7, 1907.
1904.	*Rose, George William,	September 11, 1907.
1902.	*Simpson, Gilbert Cuthbert,	June 8, 1907.
1906.	**Storey, Thomas William Patrickson,	March 15, 1907.
1902.	**Thow, Sidney,	June 24, 1907.
1879.	*Townsend, Henry Troth,	September 11, 1907.
1905.	††Variclé, Jean Antony,	July 26, 1907.
1878.	†Walker, John A.,	May 23, 1907.
1900.	*Willard, Eugene B.,	May 21, 1907.
1883.	*Williams, Lewis,	September —, 1907.
1891.	*Young, James W. R.,	February 19, 1906.

* Member. ** Life Member. † Associate. †† Life Associate.

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DEDICATION EXERCISES OF THE UNITED ENGINEERING SOCIETY BUILDING.

[NOTE.—The Committee on Publication of United Engineering Society will issue a memorial volume reporting these proceedings in full and giving a complete illustrated description of the building. For the above reason, only an outline of the dedication exercises is here presented. The following information was published more fully in *Bi-Monthly Bulletin*, No. 15, May, 1907, pp. vi to xxvi and xxxviii to lxv.]

COMMITTEES.

EXECUTIVE.—J. W. Lieb, Jr., *Chairman;* C. W. Rice, *Secretary;* John Hays Hammond, Chas. Wallace Hunt, F. R. Hutton, Charles Kirchhoff, T. C. Martin. E. E. Olcott, Chas. F. Scott, Samuel Sheldon, Ambrose Swasey.

INVITATIONS.—R. W. Raymond, *Chairman*, Charles Kirchhoff, *Secretary;* A. C. Humphreys, Robert W. Hunt, L. B. Stillwell.

PROGRAM.—A. R. Ledoux, *Chairman;* C. W. Rice, *Secretary;* Theodore Dwight, Chas. Wallace Hunt, T. C. Martin, Ralph W. Pope, Joseph Struthers.

WAYS AND MEANS.—James Douglas, *Chairman;* A. R. Ledoux, *Secretary;* H. R. Brown, J. M. Dodge, C. L. Edgar, John Hays Hammond, W. M. McFarland, E. E. Olcott, Samuel Sheldon, J. G. White.

HISTORICAL AND PUBLICATION PRESS.—T. C. Martin, *Chairman;* Albert Spies, H. H. Suplee.

RECEPTION.—J. W. Lieb, Jr., *Chairman;* C. W. Rice, *Secretary;* E. D. Adams, E. R. Archer, B. J. Arnold, J. C. Barclay, S. W. Baldwin, W. J. Baxter, N. F. Brady, J. L. Beggs, Alexander G. Bell, William P. Blake, J. A. Brashear, C. F. Brush, Andrew Carnegie, R. C. Carpenter, J. J. Carty, John A. Church, Walton Clark, Thos. F. Cole, C. A. Coffin, G. H. Condict, F. B. Crocker, A. C. Dinkey, Cleveland H. Dodge, J. M. Dodge, James Douglas, Alexander Dow, Theodore Dwight, T. A. Edison, Anton Eilers, L. A. Ferguson, J. R. Freeman, John Fritz, James Gayley, Eugene Griffen, James D. Hague, John Hays Hammond, C. H. Haswell, John B. Herreshoff, J. B. F. Herreshoff, P. C. Hewitt, C. A. Hexamer, H. M. Howe, A. C. Humphreys, Chas. Wallace Hunt, Robert W. Hunt, F. R. Hutton, W. J. Jenks, R. E. Jennings, F. W. Jones, Hon. Lord R. Kelvin, James F. Kemp, W. C. Kerr, Charles Kirchhoff, E. D. Leavitt, Albert R. Ledoux, Frank Lyman, Emerson McMillan, C. O. Mailloux, T. C. Martin, G. W. Melville, R. D. Mershon, C. H. Morgan, T. E. Murray, E. E. Olcott, F. S. Pearson, F. A. C. Perrine, R. W. Pope, Sir William Preece, M. I. Pupin, J. C. F. Randolph, R. W. Raymond, R. F. Rowland, Sir David Salomons, W. L. Saunders, C. F. Scott, Coleman Sellers, C. H. Sharp, Samuel Sheldon, Paul Spencer, C. P. Steinmetz, L. B. Stillwell, H. G. Stott, Joseph Struthers, H. H. Suplee, Ambrose Swasey, J. E. Sweet, F. H. Taylor, F. W. Taylor, C. A. Terry, P. H. Thomas, Elihu Thomson, H. R. Towne, W. R. Warner, Geo. Westinghouse, S. S. Wheeler, Arthur Williams, C. C. Worthington.

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At the first session, held in the Main Auditorium, beginning at 3 p.m. on Tuesday, April 16, 1907, Mr. Charles Wallace Hunt, past-President of the American Society of Mechanical Engineers, and representative of the Executive Committee of United Engineering Society, took the chair, and prayer was offered by Rev. Edward Everett Hale, D.D., Chaplain of the United States Senate.

Mr. T. C. Martin read a congratulatory letter from Theodore Roosevelt.

Señor Henrico Creel, Minister Plenipotentiary and Ambassador Extraordinary of the Republic of Mexico, read a personal message from General Porfirio Diaz, President of the Republic of Mexico.

A message was read from Earl Gray, Governor-General of the Dominion of Canada, who was expecting to be present, but had been prevented by an unfortunate delay of the railway train on which he was traveling.

Mr. Martin read a letter from the venerable Chas. H. Haswell, "the dean of the engineering profession," now in his 98th year, extending hearty congratulations and regretting his inability, by reason of temporary illness, to be present.

The chairman, with a few words of reminiscence, presented Mr. Charles F. Scott, past-President of the American Institute of Electrical Engineers, and "the tireless Chairman of the Joint Conference and Building Committee from its organization, four years ago, to the present time."

Mr. Scott delivered an interesting address, narrating the history of Mr. Carnegie's gift, and depicting both its present importance and its unlimited future possibilities.

The chairman, in the name of the Building Committee, delivered to Mr. E. E. Olcott, past-President of the American Institute of Mining Engineers, and President of United Engineering Society, the key of the building, which Mr. Olcott received with appropriate remarks.

[This is a Yale & Towne key, with a medallion handle, made by Tiffany & Co. of native gold from the North Star mine, Grass Valley, Cal., presented by Mr. James D. Hague, for many years President of the North Star Mining Company, and now a Vice-President of the American Institute of Mining Engineers. On one side, the medallion bears the symbols of the

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three societies. On the other side, together with appropriate ornamentation and inscription, and under a piece of quartz crystal from the Mother Lode, California, are several scales of the gold panned by John Marshall in January, 1848, from American river, California—the gift of Mr. George F. Kunz, a member of the American Institute of Mining Engineers.]

The chairman introduced Mr. Andrew Carnegie, the donor of the building, who was received with immense and long-continued applause, after which Mr. Carnegie delivered an appropriate and interesting address.*

Mr. Arthur T. Hadley, President of Yale University, then presented an admirable address setting forth the relation of men of science to the commonwealth.†

Tuesday evening was devoted to social receptions of members of the three "Founder Societies," their ladies, and invited guests. From 9 to 10.30 p. m., a general reception was held in the Main Auditorium by the Presidents of the three Founder Societies and the Chairman of the Executive Committee of United Engineering Society, with their wives, as follows:

Dr. and Mrs. John Hays Hammond, A. I. M. E.; Dr. and Mrs. Frederick R. Hutton, A. S. M. E.; Dr. and Mrs. Samuel Sheldon, A. I. E. E.; Mr. and Mrs. John W. Lieb, Jr., U. E. S.

At 10.30 p. m., a special reception was held at the rooms of each of the "Founder Societies" by the officers of the society.

The building, including the Libraries on the 12th and 13th floors, was thrown open to the inspection and admiration of guests; music was furnished on the Auditorium, fifth and Library floors; and refreshments were served during the evening in the two beautiful smaller auditoriums on the fifth floor.

At the second session, held on Wednesday afternoon at 2.30 p. m., Mr. J. W. Lieb, Jr., Chairman of the Dedication Committee, presided, and opened the exercises with an appropriate address, extending, in the name of the Founder Societies, a hearty welcome to the representatives of sister societies and

* Published in *Bi-Monthly Bulletin*, No. 15, May, 1907, p. xl.
† *Ibid.*, p. xliii.

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institutions of learning at home and abroad, and calling upon the Presidents of the Founder Societies for further remarks.

Addresses* were delivered by Dr. Samuel Sheldon, President of the American Institute of Electrical Engineers; Dr. Frederick R. Hutton, President of the American Society of Mechanical Engineers; and Dr. John Hays Hammond, President of the Council of the American Institute of Mining Engineers, and an appropriate poem was presented by Mr. T. Commerford Martin, President of the Engineers' Club.

The following societies and institutions were represented by delegates present:

REPRESENTATIVES OF INSTITUTIONS OF LEARNING.

Brown University, Professor William H. Kenerson.

Case School of Applied Science, President Charles S. Howe.

Thomas S. Clarkson Memorial School of Technology, Director C. Aldrich.

University of Cincinnati, President Charles W. Dabney and Dr. Thomas Evans.

Colorado College, President William F. Slocum.

Colorado School of Mines, Arthur R. Townsend.

Columbia University, Frederick A. Goetze, and Professor William H. Burr.

Cornell University, Walter C. Kerr.

University of Georgia, Professor C. M. Strahan.

Haverford College, Professor L. H. Rittenhouse.

Johns Hopkins University, R. Brent Keyser.

University of Illinois, E. W. Goldschmidt.

State College of Kentucky, President James K. Patterson.

Lafayette College, President E. D. Warfield.

Lehigh University, President Henry S. Drinker.

Massachusetts Institute of Technology, Professor R. H. Richards.

University of Michigan, Alfred Noble.

Michigan Agricultural College, Joseph T. Freeman.

United States Military Academy, Captain Richmond P. Davis.

University of Minnesota, Professor Fred. S. Jones.

University of Missouri, Charles Lewis Harrison.

* Published in *Bi-Monthly Bulletin*, No. 15, May 1907, p. lii.

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United States Naval Academy, Commander John K. Barton.

College of the City of New York, Professor Albert G. Compton.

New York University, Professor Collins P. Bliss and Professor Charles H. Snow.

Ohio University, President Alston Ellis.

University of Pennsylvania, Professor Henry Wilson Spangler.

Western University of Pennsylvania, Chancellor S. B. McCormick.

Polytechnic Institute of Brooklyn, President Fred. W. Atkinson.

Pratt Institute, Director Arthur L. Williston.

Princeton University, Professor Chas. McMillan.

Rensselaer Polytechnic Institute, President P. Ricketts.

Rutgers College, Professor Alfred A. Titsworth.

Stevens Institute of Technology, James E. Denton and Dr. D. S. Jacobus.

St. John's College, President Thomas Fell.

Trinity College, Professor Henry Augustus Perkins.

Tufts College, Professor Gardner C. Anthony.

Union University, Professor Olin H. Landreth.

University of Vermont, Professor W. H. Freedman.

Washington and Lee University, President George H. Denny.

George Washington University, President Charles Willis Needham.

University of Washington, Professor H. K. Benson.

Worcester Polytechnic Institute, Professor L. P. Kinnicutt.

Yale University, President Arthur T. Hadley.

REPRESENTATIVES OF ENGINEERING AND SCIENTIFIC SOCIETIES.

American Ceramic Society, Stanley G. Burt, President.

American Chemical Society, Marston T. Bogert, President.

American Electrochemical Society, Samuel A. Tucker, Chairman, and Dr. E. F. Roeber.

American Foundrymen's Association, Dr. Richard Moldenke, Secretary.

American Gas Institute, Henry L. Doherty.

American Institute of Architects, Cass Gilbert.

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American Railway Associations, W. F. Allen, Secretary and Treasurer.

American Railway Master Mechanics' Association, Arthur M. Waitt, past-President.

American Society of Civil Engineers, Nelson P. Lewis.

American Society of Heating and Ventilating Engineers, W. M. Mackay, Secretary.

American Society of Municipal Improvements, George W. Tillson, Secretary.

American Society of Naval Engineers, Commander Albert Moritz.

American Society of Refrigerating Engineers, John E. Starr, President.

American Society for Testing Materials, Professor Edgar Marburg, Secretary-Treasurer.

American Water Works Association, Morris R. Sherrerd.

Association of Railway Superintendents of Bridges and Buildings, Joseph H. Cummin.

Associazione Elettrotecnica Italiana, J. W. Lieb, Jr.

Boston Society of Civil Engineers, Francis W. Dean, Vice-President.

Brooklyn Engineers' Club, C. E. Pollock, President.

Canadian Society of Civil Engineers, W. McLea Walbank.

Concrete Association of New York, Albert Mayer.

Corpo Reale Dellé Miniere, Dr. R. W. Raymond.

Deutsche Chemische Gesellschaft, Dr. C. F. Chandler.

Electrical Contractors' Association, James Hilton.

Empire State Gas and Electric Association, T. R. Beal, Treasurer.

Engineers' Club, T. Commerford Martin, President.

Engineers' Society of Western New York, Harry B. Alvorson.

Faraday Society, Leon Gaster.

Geological Society of America, Prof. J. J. Stevenson.

Illuminating Engineering Society, Dr. Clayton H. Sharp, President.

Institution of Electrical Engineers, Sir William Preece.

Institution of Mechanical Engineers, Ambrose Swasey.

Institute of Marine Engineers, N. K. McLean.

Iron and Steel Institute, R. A. Hadfield, President.

Italian Society of Engineers and Architects, J. W. Lieb, Jr.

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Koninklijk Instituut Van Ingenieurs, F. W. Ruhle von Lilienstern ter Meulen.

Master Car Builders' Association, F. W. Brazier.

Municipal Engineers of New York, George S. Rice, President.

National Association of Cotton Manufacturers, F. A. Flather.

National Electric Light Association, Arthur Williams, President.

National Fire Protection Association, Charles A. Hexamer, President.

New England Association of Gas Engineers, William McGregor, President.

New England Water Works Association, M. N. Baker, Vice-President.

New York Electrical Society, G. Herbert Condict, President; G. H. Guy, Secretary; H. A. Sinclair, Treasurer.

New York Railroad Club, W. G. Besler, Vice-President.

North of England Institute of Mining and Mechanical Engineers, J. Parke Channing.

Österreicher Ingenieur- und Architekten-Verein, George W. McNulty.

Philosophical Society of Washington, G. R. Putnam.

Railway Signal Association, A. H. Rudd, Vice-President.

Society of Automobile Engineers, A. L. Riker, President.

Society of Beaux-Arts Architects, Lloyd Warren, President.

Society of Chemical Industry, T. J. Parker.

Société des Ingenieurs Civils de France, Sorzano de Tejada.

Society of Arts, Sir William Preece.

Society of Naval Architects and Marine Engineers, Stevenson Taylor, Vice-President.

Technical Society of the Pacific Coast, G. W. Dickie, past-President.

Trustees Gas Educational Fund, W. R. Beal, Treasurer.

Verein deutscher Eisenhüttenleute, C. Kirchhoff.

Western Society of Engineers, Professor Morgan Brooks.

The time being manifestly too brief to permit the hearing of written or oral addresses from all these distinguished representatives, the chairman called upon the following few only, who responded in brief but happy messages of congratulation:

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Sir William Henry Preece, for the Institution of Electrical Engineers of Great Britain.

Mr. Walter C. Kerr, Trustee of Cornell University, for American universities.

Mr. R. A. Hadfield, President of the Iron and Steel Institute of Great Britain, for that society. (In addition to his cordial and graceful speech, Mr. Hadfield presented an illuminated address from the Iron and Steel Institute.)

Dr. Friedrich Eichler, for the Verband deutscher elektrischer Techniker. (This address was delivered in German.)

Mr. Sorzano de Tejada, for the Society of Civil Engineers of France.

Mr. John F. Wallace, past-President of the American Society of Civil Engineers, who, in the unavoidable absence of President G. H. Benzenberg of that society, presented a cordial communication from the latter.

Dr. Henry Pritchett, President of the Carnegie Foundation, Washington, D. C., and past-President of the Massachusetts Institute of Technology, for the latter institution and technical schools generally.

Mr. Charles Kirchhoff, for the Verein deutscher Eisenhüttenleute. (Mr. Kirchhoff presented an illuminated address.)

Mr. Carl Hering, past-President of the American Institute of Electrical Engineers, for the Société Internationale des Electriciens, of Paris, France.

Mr. Leon Gaster, for the Faraday Society of Great Britain.

Dr. F. R. Archenhold, Director of the Royal Prussian Astronomical Observatory at Treptow, Berlin, presented the greeting of that institution.

Capt. Baxter, U. S. N., President of the Society of Naval Architects and Marine Engineers, spoke for the Associate Societies occupying rooms in the new building.

The Chairman presented a large number of congratulatory letters and telegrams, received from the following:

Lord Kelvin; the Institution of Civil Engineers; the Institution of Mining and Metallurgy; the French National Conservatory of Arts and Sciences; the French International Bureau of Weights and Measures; and many others.

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The John Fritz Medal.

Mr. Charles F. Scott, President of the Board of Award of the John Fritz medal, conferred the medal, in the name of the Board, upon Dr. Alexander Graham Bell, for his invention of the telephone.

Dr. Bell responded in a few hearty words, concluding, amid great applause, with the declaration that this honor was made still more precious to him by the fact that it was bestowed in the presence of the venerable John Fritz himself.

Commemorative Secretaries' Medals.

Dr. A. R. Ledoux, past-President of the American Institute of Mining Engineers and of United Engineering Society, was then called upon to present to Dr. F. R. Hutton, past-Secretary of the American Society of Mechanical Engineers; Mr. Ralph J. Pope, Secretary of the American Institute of Electrical Engineers; and Dr. R. W. Raymond, Secretary of the American Institute of Mining Engineers, three gold medals, severally bestowed by the three societies, in recognition of the long service of these officers. Dr. Ledoux prefaced each of these presentations with an appropriate sketch of the career of the recipient.

Dr. Hutton replied for all three Secretaries.

The Building.

The illustrated description of the United Engineering Society Building, No. 29 West 39th Street, New York, N. Y., prepared for the Dedication Exercises by the Historical and Publication Press Committee, Mr. T. Commerford Martin (*Chairman*), Mr. Albert Spies and Mr. H. H. Suplee, was published in *Bi-Monthly Bulletin*, No. 15, May, 1907, pp. vi. to xxvi.

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Proceedings of the Ninety-Second Meeting, New York, N. Y., April, 1907.

THIS meeting was held in the new home of the Institute, the United Engineering Society Building, 29 West 39th Street, New York, N. Y., directly following the Dedication ceremonies.

The first session was held in the large Auditorium, Thursday, April 18.

The meeting was called to order by Mr. John Hays Hammond, President of the Council, who introduced Dr. Rossiter W. Raymond, who delivered an address in his usual felicitous style, outlining the history of the organization and the development of the Institute.*

The following paper, illustrated by lantern-views, was presented in oral abstract by the author:

Mining Operations in New York City and Vicinity, by H. T. Hildage, New York, N. Y.

The second and concluding session was held Friday, April 19, at 2.30 p.m., in the large assembly-room, President Hammond presiding.

The following papers were presented in oral abstract by the authors:

The Influence of the Conditions of Casting on Piping and Segregation, as Shown by Means of Wax Ingots, by H. M. Howe and Bradley Stoughton, New York, N. Y. (Discussed by Robert W. Hunt, Chicago, Ill.)

The White Knob Copper-Deposits, Mackay, Idaho, by James F. Kemp, New York, N. Y., and C. G. Gunther, Clifton, Ariz.

Laboratory Experiments in Lime-Roasting a Galena-Concentrate with Reference to the Savelsberg Process, by H. O. Hofman, R. P. Reynolds and A. E. Wells, Boston, Mass. (Prof. Hofman in the absence of the author read, in oral abstract, and replied to the discussion of his paper, by George A. Packard, Rolla, Mo.)

Discussion of Paper of Mr. Meissner, Notes on the Gayley

* Published in *Bi-Monthly Bulletin*, No. 15, May, 1907, p. 541.

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Dry-Air Blast-Process, by J. E. Johnson, Jr., Glen Wilton, Va. (Discussed by Mr. Meissner, Prof. Howe and Mr. Johnson.)

The following papers were presented in printed form:

Grinding in Tube-Mills at the Waihi Gold-Mine, Waihi, New Zealand, by E. G. Banks, Waihi, New Zealand.

The Butters Slime-Filter at the Cyanide Plant of the Combination Mines Company, Goldfield, Nev., by Mark R. Lamb, Goldfield, Nev.

Fluorite and Barite in Tennessee, by Thomas L. Watson, Blacksburg, Va.*

Geology of the Exposed Treasure Lode, Mojave, Cal., by Courtenay De Kalb, Los Angeles, Cal.

Deutschman's Cave, near Glacier, B. C., Canada, by W. S. Ayres, Banff, Alberta.

The Constitution of Ferro-Cuprous Sulphides, by H. O. Hofman, W. S. Caypless and E. E. Harrington, Boston, Mass.

The Roasting of the Argentiferous Cobalt-Nickel Arsenides of Temiskaming, Ontario, Canada, by H. M. Howe, New York, N. Y.; William Campbell, New York, N. Y.; and C. W. Knight, New York, N. Y.

Relative Elimination of Iron, Sulphur, and Arsenic in Bessemerizing Copper-Mattes, by E. P. Mathewson, Anaconda, Mont.

Piping and Segregation in Steel Ingots, by H. M. Howe, New York, N. Y.

An Early Instance of Blowing-In without "Scaffolding Down," by Frank Firmstone, Easton, Pa.

The Extraordinary Faulting at the Berlin Mine, Nevada, by Ellsworth Daggett, Salt Lake City, Utah.

The Ore-Deposits of the Joplin Region, Missouri, by F. L. Clerc, Denver, Colo.

A Study in Refining and Overpoling Electrolytic Copper, by H. O. Hofman, R. Hayden and H. B. Hallowell, Boston, Mass.

The Formation and Enrichment of Ore-Bearing Veins, by George J. Bancroft, Denver, Colo.

Search for the Causes of Injury to Vegetation in an Urban Villa near a Large Industrial Establishment, by Persifor Frazer, Philadelphia, Pa.

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* *Trans.*, xxxvii., 890.

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Bibliography of Injuries to Vegetation by Furnace-Gases, by Persifor Frazer, Philadelphia, Pa.

The Velocity of Galena and Quartz Falling in Water, by Robert H. Richards, Boston, Mass.

Discussion of Paper of Messrs. Gibb and Philp, The Constitution of Mattes Produced in Copper-Smelting, by Allan Gibb, Queensland, Australia.

Discussion of Paper of Mr. Read, The Secondary Enrichment of Copper-Iron Sulphides, by Eugene C. Sullivan, Washington, D. C., and Reply by T. T. Read, Colorado Springs, Colo.*

Discussion of Paper of Mr. Colby, Comparison of American and Foreign Rail-Specifications, with a Proposed Standard Specification to Cover American Rails Rolled for Export, by E. Windsor Richards, London, England; R. Price-Williams, London, England; F. W. Harbord, London, England; R. A. Hadfield, London, England; J. E. Stead, Middlesbrough, England; James E. York, New York, N. Y.; A. Lamberton, Sheffield, England; Robert W. Hunt, Chicago, Ill.; E. F. Kenney, Philadelphia, Pa.; W. E. Freir, London, England; William R. Webster, Philadelphia, Pa.; C. S. R. Palmer, London, England; Albert Sauveur, Cambridge, Mass.; M. Nigoud, Paris, France;† and Reply by Albert Ladd Colby, New York, N. Y.

Discussion of Papers of Messrs. Hubert, Westgarth and Reinhardt on Gas-Engine Practice, by Adolph Greiner, Seraing, Belgium; Tom Westgarth, Middlesbrough, England; Julian Kennedy, Pittsburg, Pa.; R. W. Raymond, New York, N. Y.; William Kent, Syracuse, N. Y.; E. J. Duff, Liverpool, England; James Hamilton, Coatbridge, England; A. T. Tannett-Walker, Leeds, England; Mark Robinson, London, England; Professor Turner, Birmingham, England; and B. H. Thwaite, London, England.‡

Discussion of Paper of Mr. York, Improvements in Rolling Iron and Steel, by R. W. Hunt, Chicago, Ill., and Kurt Kerlen, Düsseldorf, Germany.§

Discussion of Paper of Mr. Grammer, Flue-Dirt and Top-Pressure in Iron Blast-Furnaces: a Study of the Influences Controlling Them, and the Paper of Mr. Johnson, Physical Action of the Blast-Furnace, by T. F. Witherbee, Durango, Mexico.

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* *Trans.*, xxxvii., 893.

† *Trans.*, xxxvii., 900.

‡ *Trans.*, xxxvii., 924.

§ *Trans.*, xxxvii., 896.

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Discussion of Paper of Mr. Corson, Heat-Treatment of Steels Containing Fifty Hundredths and Eighty Hundredths Per Cent. of Carbon, by Albert Sauveur, Cambridge, Mass.*

Discussion of Paper of Messrs. Hofman, Reynolds and Wells, Laboratory Experiments in Lime-Roasting a Galena-Concentrate, by George A. Packard, Rolla, Mo.

Discussion of Paper of Mr. Lee, The Gas-Producer as an Auxiliary in Iron Blast-Furnace Practice, by J. T. Pullon, Leeds, England; F. T. Havard, Anhalt, Germany; and William Kent, Syracuse, N. Y.†

The following papers were presented in manuscript:

Barite Associated with Iron-Ore in Pinar del Rio Province, Cuba, by Charles Catlett, Staunton, Va.

The Presence of Gold and Silver in Deep-Sea Dredgings, by Luther Wagoner, San Francisco, Cal.

The Vein-System of the Standard Mine, Bodie, Cal., by R. Gilman Brown, San Francisco, Cal.

The South-African Tin-Deposits, by William R. Rumbold, Oruro, Bolivia, South America.‡

Chlorination of Gold-Ores; Laboratory-Tests, by A. L. Sweetser, Cananea, Sonora, Mexico.

The Verschoyle Pocket Transit, by W. D. Verschoyle, Ballisodare, Ireland.

Biographical Notice of Thomas Septimus Austin, by A. S. Dwight, New York, N. Y.

Biographical Notice of William George Neilson, by John Birkinbine, Philadelphia, Pa.

Discussion of Paper of Mr. Mathewson, Relative Elimination of Iron, Sulphur, and Arsenic in Bessemerizing Copper-Mattes, by H. M. Howe, New York, N. Y.

Discussion of Paper of Mr. Howe, Piping and Segregation in Steel Ingots, by F. Beutter, Paris, France.

Excursions and Entertainments.

An account of the excursions and entertainments in which the members and guests of the Institute participated was published in *Bi-Monthly Bulletin*, No. 15, May, 1907, p. 552.

* *Trans.*, xxxvii., 936. † *Trans.*, xxxvii., 920.

‡ Proof not received from author in time for publication in this volume.

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List of Members and Guests Registered at New York.

(*Probably incomplete.*)

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Proceedings of the Ninety-Third Meeting, Toronto, Canada, July, 1907.

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The first session, held in the Banquet Hall of the King Edward Hotel, was called to order Tuesday, July 23, 1907, at 2.30 p.m., President John Hays Hammond presiding.

Prof. Willett G. Miller, chairman of the General Committee, introduced Alderman J. J. Graham, who, in behalf of the City of Toronto, extended a hearty welcome to the members and guests of the Institute. To this welcome, President Hammond replied in a few well-chosen and hearty remarks.

In the hall in which the session was held there was exhibited a fine, full-length, life-size portrait of Dr. Rossiter W. Raymond, in 1871 Vice-President and acting President, in 1872, 1873 and 1874 the elected President, in 1876 and 1877 Vice-President, in 1883, by appointment of the Council, and for the last 24 years by annual election, Secretary, of the Institute.

Dr. James Douglas, of New York, N. Y., on behalf of numerous members, presented this portrait to Dr. Raymond, with a highly cordial and complimentary address.

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Dr. Raymond, after acknowledging with gratitude and pride this testimonial of affection and approval, announced that he would immediately present the portrait to the Institute, in the rooms of which he hoped it might serve, after he had departed, to recall his face and bear witness of the friendly and flattering estimate which had been placed upon his work.

The Secretary announced that, upon the proposal of many members (including nearly all the past-Presidents of the Institute now living), the unanimous report of the Committee on Membership, and the unanimous recommendation of the Council, Dr. Charles D. Walcott, late Director of the U. S. Geological Survey and now Secretary of the Smithsonian Institution, had been unanimously elected by the Board of Directors an Honorary Member of the American Institute of Mining Engineers, in recognition of his eminent professional contributions and official services to the science of geology, and of his cordial and efficient co-operation with the work of the Institute.

The following papers were presented in oral abstract by the authors:

The Destruction of the Salt-Works in the Colorado Desert by the Salton Sea, by William P. Blake, Tucson, Ariz.

Secrecy in the Arts, by Dr. James Douglas, New York, N. Y.

The second session was held at the same place on Wednesday, July 24, beginning at 10.30 a.m., President Hammond in the chair.

The Secretary, in the absence of the author, presented in oral abstract the following paper:

The Corrosion of Water-Jackets of Copper Blast-Furnaces, by George B. Lee, Douglas, Ariz.

The following papers were presented in oral abstract by the authors:

The Electric-Air Drill, by William L. Saunders, New York, N. Y. Mr. Saunders's paper, which was illustrated by lantern-views, brought forth an animated and interesting discussion by Messrs. D. B. Rushmore, Thos. J. Eynon, Wm. Kent, E. W. Parker, R. W. Raymond and W. L. Saunders.

Coal-Briquetting in the United States, by Edward W. Parker, Washington, D. C.

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The Tar-Sands of the Athabasca River, Canada, by Dr. Robert Bell, Ottawa, Can. Dr. Bell's paper was discussed by Mr. Eugene Coste.

The third and concluding session, held at the same place, on Wednesday, July 24, was called to order, 2.30 p.m., by President Hammond.

The following papers, illustrated by lantern-views, were presented in oral abstract by the authors:

Notes on the Cobalt Mineral-Area, by Willett G. Miller, Toronto, Ont.*

Notes on the Sudbury Mineral-Area, by Dr. Alfred E. Barlow, Ottawa, Ont.*

The following papers were read by title for future publication by the Institute:

The Wilfley Table, I., by R. H. Richards, Boston, Mass.

The Occurrence of Nickel in Virginia, by Thomas L. Watson, Blacksburg, Va.

Geology of the Virginia Barite-Deposits, by Thomas L. Watson, Blacksburg, Va.

The Effect of High Litharge in the Crucible-Assay for Silver, by Richard W. Lodge, Boston, Mass.

Physical Factors in the Metallurgical Reduction of Zinc Oxide, by Woolsey McA. Johnson, New York, N. Y.

Zinc Oxide in Iron-Ores, and the Effect of Zinc in the Iron Blast-Furnace, by John J. Porter, Cincinnati, Ohio.

Chronology of Lead-Mining in the United States, by Walter R. Ingalls, New York, N. Y.

The Promontorio Silver-Mine, Durango, Mexico, by Francis C. Lincoln, New York, N. Y.

Blow-Holes in Steel Ingots, by E. von Maltitz, South Chicago, Ill.

The Evergreen Copper-Deposit, by Etienne A. Ritter, Colorado Springs, Colo.

Pure Coal as a Basis for the Comparison of Bituminous Coals, by W. F. Wheeler, Urbana, Ill.

Quantitative Field-Test for Magnesia in Cement-Rock and Limestone, by Charles Catlett, Staunton, Va.

The Production of Converter-Matte from Copper-Concen-

* Not furnished for publication.

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trates by Pot-Roasting and Smelting, by George A. Packard, Boston, Mass.

Biographical Notices of 1906.*

The Panoramic Camera Applied to Photo-Topographic Work, by Charles W. Wright, Washington, D. C.

The Present Source and Uses of Vanadium, by J. Kent Smith, Pittsburg, Pa.

Discussion of Paper of H. M. Howe on Piping and Segregation in Steel Ingots, by Alfred C. Lane, Lansing, Mich.

Geology and Mining of the Tin-Deposits of Cape Prince of Wales, Alaska, by Albert Hill Fay, New York, N. Y.

The Geological Relations of the Scandinavian Iron-Ores, by Hjalmar Sjögren, Stockholm, Sweden.

The Ore-Deposits of the Eastern Gold-Belt of North Carolina, by W. O. Crosby, Boston, Mass.

Excursions and Entertainments.

An account of the excursions and entertainments in which members of the Institute and their guests participated was published in *Bi-Monthly Bulletin*, No. 17, September, 1907, pp. 863 to 875.

List of Members and Guests Registered at Toronto or Attending, in Whole or in Part, the Train-Trip to Cobalt and Return.

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* Published in *Bi-Monthly Bulletin*, No. 16, July, 1907, p. 657.

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PAPERS.

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Piping and Segregation in Steel Ingots.

PRELIMINARY PAPER.

BY HENRY M. HOWE, LL.D., NEW YORK, N. Y.*

(London Meeting, July, 1906.)

§ 1. *Introduction.*—The first part of this paper[a] treats of the causes and the restraining of piping in steel ingots; the second[b] considers the causes and the restraining of segregation; and the third[c] proposes certain precautions in engineering specifications concerning these two defects. Into their causes I have looked carefully, with the purpose of arriving at rational methods of restraining them, and at efficient ways of detecting and measuring the harm which they cause, so as to benefit the public by reducing this harm at once effectively and with the least expense to the manufacturer, and therefore in the end with the least expense to the consumer.

In Part I., I infer that the pipe is chiefly due to what I call the virtual expansion of the outer walls of the ingot in the early part of the freezing.[d] I find that the upper and smooth-faced part of the pipe probably forms while the interior is still molten, but that the lower, steep, and crystal-faced part probably forms in metal which is already firm. Of the five causes which may co-operate to limit the depth of the pipe, I find that three—blow-holes, sagging, and the progress of freezing from below upwards—are usually effective. I find that the pipe may be lessened by casting (1) in wide ingots[e]; (2) in sand molds,[f] especially if these are pre-heated (this is rarely expedient); (3) at the top[g] instead of at the bottom; (4) slowly[h]; (5) and with the large end up[i]; (6) by the use of a sinking-head or other means of retarding the cooling of the top[j]; (7) by permitting blow-holes to form[k]; and (8) by liquid compression.[l] Most of these I consider in some detail, and in particular I dwell on the advantages of casting with the large end up, and I propose certain administrative arrangements to permit this.

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* Professor of Metallurgy, Columbia University, New York, N. Y.

[a] P. 5. [b] P. 75. [c] P. 99. [d] P. 17. [e] P. 56. [f] P. 57.
[g] P. 59. [h] P. 60. [i] P. 60. [j] P. 66. [k] P. 70. [l] P. 70.

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In Part II., p. 75, I find that, although the reasons why (1) casting in wide ingots and (2) in sand- or clay-lined molds shorten the pipe do not apply to show that they should raise the segregate, yet the position of the segregate should be raised by the six other means by which the pipe is shortened, viz.: by casting (3) at the top, (4) slowly, and (5) with the large end up; (6) by the use of a sinking-head or other means of retarding the cooling of the top; (7) by allowing blow-holes to form; and (8) by liquid compression.

I next consider in § 53, p. 77, the means proposed for lessening the degree of segregation, as distinguished from raising the position of the segregate, viz.:

(9) Quieting the steel by adding aluminum;

(10) Casting in small instead of in large ingots; and hastening the solidification, not only by casting in small ingots, but also

(11) By casting at a low temperature;

(12) By casting in thick-walled iron molds (*i.e.*, those of high thermal conductivity); and

(13) By casting slowly.

Pending the completion of further experiments and an analysis of the data, I point out that (9) quieting the steel has materially lessened segregation in certain cases, and I hold that (10) segregation is probably much less in small than in large ingots, but that rapid cooling has certain effects which tend to lessen segregation and others which tend to increase it, so that its net effect should be expected to differ both in importance and in sign from case to case; and I find that the evidence agrees with this inference.[1]

[1] In the original draft of this paper, prepared for the London meeting under more pressure than favors proper deliberation, I adopted, with some qualifications, the current opinions that slow cooling increases segregation and rapid cooling opposes it, and that segregation is much greater in large than in small ingots. Prolonged further study of the conditions and evidence, while it goes to show that my contention as regards ingot-size was right, yet shows that in sudden cooling two sets of causes are at work, of which one tends to lessen and the other to increase segregation. But the influence of ingot-size and that of rate of cooling are so important that I have decided to seek further and more conclusive evidence by means of direct experiments, which are now in hand. Their results, together with a thorough analysis of the present data, I hope to present in a later paper. In 1894 I expressed my belief "that there are many other cases in which hastening solidification favors segregation." (*Journal of the Iron and Steel Institute*, vol. xlvi. (1894, No. II.), p. 115.)

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I consider at great length in § 60 the effectiveness of the different methods of fluid compression, concluding that Whitworth's system should be the least effective, S. T. Williams's the most effective, and the systems of Harmet and Illingworth intermediate in effectiveness in raising the segregate.

In Part III., I consider briefly the relations of maker and buyer, the reasons which lead to secrecy in manufacture, and especially the imperfect protection which patents can give to many metallurgical inventions. I then lay stress on inspection at the rolls and shears, and especially on axial drilling of the billets or other products.

The different means of restraining piping and segregation which have here been studied are then recapitulated,[m] and the paper ends with an explanation of why it is that, of these various means, only (3) top casting can well be insisted on in the majority of cases; and why in certain special important cases the buyer may consider carefully whether he may not reasonably insist that casting shall be done (5) with the large end up, (6) with a sinking-head or its equivalent, and (10) in ingots not larger than 8 in. square.

I. Piping.

§ 2. *When does the pipe form?* The pipe in a cold ingot, *E*, Fig. 1, is a hollow space, filled with gas, and shaped like an inverted bell. This shape goes to show that the pipe forms during solidification, somewhat as sketched in Fig. 1, although we shall see in §§ 32 and 33 that, when its lower end stretches down with steep, nearly parallel, and rough sides, this lower part is probably opened as a chasm in the already solid metal. Let us now follow the general course of the formation of the pipe.

When the very outer crust of the ingot solidifies, its form and size are those of the molten metal within it; it fits over this molten metal as a bottle does over its contents of water, or as a tight glove fits over the hand. As freezing proceeds, and the solid walls of the ingot grow thicker by the deposition of successive layers of solid steel out of the molten interior, layer upon layer, a moment arrives when this molten interior no longer suffices to fill completely the solid inclosing crust, somewhat as if during the progress of emaciation from some protracted illness, I should retain the glove which once fitted my hand

[m] P. 106.

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but now creases over it, or as if water were to leak out through a crack in the bottom of a once well-filled bottle. As the water leaks out more and more, the tide in the bottle may be said to fall gradually. As the ebbing tide leaves on the beach a deposit, of which each line represents the water-mark at some given instant in that ebb, so if the water is soiled it leaves on

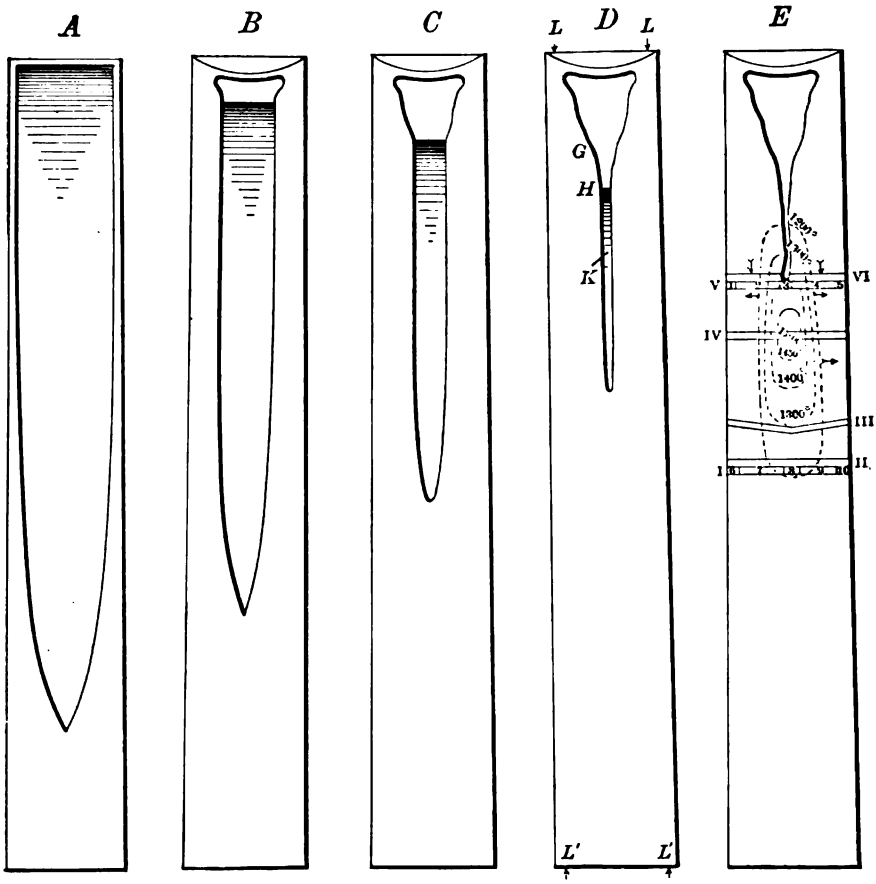

FIG. 1.—SUPPOSED GENESIS OF A PIPE IN A SOLIDIFYING STEEL INGOT.

the sides of our bottle such a succession of lines, representing the level of the water's surface at successive stages during this *quasi* ebb; and thus a succession of imaginary horizontal lines around the surface of the pipe represents the upper surface of the molten steel at successive stages during the ebb of this land-bound, indeed subterranean, deep lake or covered well, the progressively sinking mass of molten steel.

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Fig. 2 shows on a larger scale the condition of affairs at a single stage during the formation of the pipe, when the tide has already ebbed from *ef* to *bg*, leaving the space *efbg* vacant. Below the surface *bg* the lake of molten steel penetrates deep down into the ingot, with the outer walls of which its sides are roughly parallel: although they probably draw together in the lower part somewhat as sketched in Fig. 1, first because the steel in the lower part is poured into the mold earlier than that in the upper part, so that at any given instant cooling and solidification have advanced further in the lower than in the upper part; and second, because the cooling of the lower part is hastened by its necessarily firm contact with the thick and initially cold stool which forms the bottom of the mold.

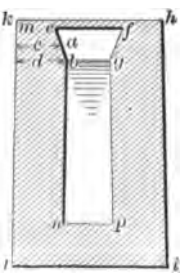

FIG. 2.—SUPPOSED GENESIS OF A PIPE IN A FREEZING STEEL INGOT.

The diameter of the pipe at any given level, say *a* in Fig. 2, represents approximately the width of the upper surface of the subterranean molten lake at the moment when this was at this same level.[2] The diameter of the pipe at a somewhat lower level, *b*, gives the width of this lake when the tide had fallen to *b*; the thickness *c* represents the thickness to which the walls had frozen when the tide was at level *a*, and the thickness *d* represents that of the walls when the tide had fallen to *b*; and the excess of *d* over *c* represents the amount of thickening of the walls which took place while the tide was sinking from *a* to *b*.

Thus the thickness of the ingot-walls, *em*, at the very top of the pipe is the thickness of those walls when the molten metal first ceased to fill completely the solid shell, when the first incipiency of the pipe formed, when the tide first began to fall; and the fact that the pipe decreases in width from top to bottom shows that while it was forming—*i.e.*, while the tide was gradually falling—the ingot-walls were gradually thickening; or, in short, that the pipe forms during freezing, while the steel is passing progressively from the molten to the solid state. This might seem clear enough beforehand; but I want to leave

[2] This width is, of course, modified by the contraction of the metal during the subsequent cooling; but throughout this section I ignore these later modifications of size for simplicity of presentation.

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no doubt in your mind that it is during this freezing of the metal, during the successive deposition of solid layer after layer from out of the molten lake against its shores, the already solid walls, that the pipe actually forms.

The sharp pointing of the lower end of the pipe indicates that it continued to open at least up to the time when the last remnant of a puddle there had frozen; for otherwise this lower end would be level, or at least meniscus shaped, like the surface of water frozen at the bottom of a conical mold.

The shape and position of the pipe in the zinc ingot, shown in Fig. 15, III., p. 41, leave hardly a doubt that here the pipe has formed in the way just described. When we compare this pipe with those in ingots I. and II. of this same figure, we readily admit that these, too, have probably formed in this same way, and the more readily after we have seen in § 42 that casting these with their large ends down ought to have just this effect of very greatly lengthening the pipe.

From this admission to the further one that this is the way in which the usual subterranean pipe is formed in steel ingots, such as are shown in Figs. 1, 2, 12, 13, 20, 21, and 22, is an easy step, made easier by the fact that even in steel the pipe is open at the top and in general like that of Figs. 15, I. and II., when the conditions favor this form, as will be explained in § 38.

§ 3. *Why does the pipe form?* Our common answer has been substantially this: "The dimensions of the shell of the ingot at the moment when it solidifies are determined by the volume of the molten interior. The contraction of this interior in freezing exceeds the contraction of the shell, progressively more and more, so that the molten interior falls progressively further and further short of filling completely its inclosing shell; and consequently the empty space or pipe in the cold ingot represents this shortage."

§ 4. *Metals which expand in solidifying also contain a pipe.* The foregoing answer, though it contains much truth, not only is inaccurate in form, but wholly fails to explain either why gray cast-iron, which certainly expands in solidifying, also forms a pipe,[3] or why the pipe is so large in steel ingots.

[3] In casting large cast-iron rolls, which stand with their axis vertical when casting, unless they are properly fed to anticipate this piping tendency the pipe itself may stretch down as much as a foot, and there may be a spongy region

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Fig. 1, *E*, shows an ingot in my collection with a pipe which seems very much too great to be due simply to the excess of contraction of the molten interior over that of the outer shell. When the very outer shell is freezing, the molten interior must be very near the freezing-point; there cannot be any large difference of temperature between them. That the contraction of the molten interior in cooling slowly through these few degrees to the freezing-point should exceed the simultaneous contraction of the now fast-cooling shell by the amount which this large pipe represents, seems to me simply inconceivable.

Because this explanation fails to explain either the presence of a pipe in gray cast-iron, or why the pipe is at times so large in steel ingots, and, indeed, in ingots in general, there must be another cause for the pipe, and this cause I will explain in § 6.

§ 5. *Does steel expand or contract in solidifying?* Does it expand as water[4] and gray cast-iron do, or does it behave like the great majority of substances and contract progressively in cooling towards, at, and below the freezing-point, until the critical temperatures (Ar_3, etc.) of the solid state are reached?

When I began this study I certainly believed that iron expanded, because I was so familiar with the floating of the puddled bars which I used to charge in the acid open-hearth furnace. In my own practice these puddled bars were pre-heated, and they most certainly floated about in the bath, and as I remember them they

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stretching down another foot below the bottom of the pipe. (Richard Moldenke, Ph.D., private communication, May 15, 1906.) Naturally, there is less piping in gray than in white cast-iron, because the separation of graphite in the former causes an expansion which tends to efface the pipe.

[4] Those who see in evolution not aimless chance but benevolent design, a predesigning of the world to fit a man who, after endless æons, would inhabit it, may point to the floating of ice as a benevolent way, first of preventing ponds from freezing solid to the bottom, and therefore remaining ice-cold until midsummer, and second, of giving man a bridge across stream and lake at a season when his body could not endure swimming, wading, or even fording. They can point to the sudden expansion in freezing of the most important of all casting-metals, gray cast-iron, as a benevolent means of enabling man to fill his molds sharply. They can point further to the fact, if, as I believe, it is a fact, that man's eye can read temperature by the color of red-hot steel incomparably more accurately in that range in which he needs great accuracy—viz., in the range in which steel acquires the hardening power—than in any other range of temperature. Here the eye seems to have been pre-fitted to a special purpose, for which it was not to be used until geological ages after that pre-fitting.

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floated evenly, like so many boards in water. On consulting others of long experience with the open-hearth process I find that most of them are positive about this floating; indeed, the reason why I took up this study was that I could not answer on the spot the question put to me by an eminent open-hearth steel-maker, "How comes it that, though solid steel is certainly lighter than molten steel, as is shown by its floating, yet in freezing a pipe forms within it?"

Further study has led me to doubt, for four reasons, whether solid steel is really lighter than the same steel when molten: (1) a very competent observer with admirable opportunities informs me that, in his basic open-hearth practice, solid scrap-steel never floats on the molten bath except under unusual conditions; (2) when I had pieces of hammered steel about 2 in. square and 8 in. long immersed in the molten steel in the casting-ladle after the slag had been pushed aside, they invariably sank out of sight and did not reappear, even in a case in which the solid steel had been strongly pre-heated; (3) Moissan's observation that, though buttons of cast-iron nearly saturated with carbon, in freezing eject molten matter through their upper crust, yet like buttons of steel containing less than 1 per cent. of carbon never do, led him to infer that, though such cast-iron expands in solidifying, steel contracts;[5] (4) the fact that the segregate in steel ingots is invariably much above the middle of the length of the ingot, goes to show that molten steel continues contracting as it nears the freezing-point, as explained in § 46, p. 68.

If we were forced to make a definite assumption, then, in view of the fact that there is no conclusive proof that steel expands, and that the circumstantial evidence is inconclusive, pointing about as strongly in one direction as in the other, our reasonable course would be to assume that steel follows the general course of nature and contracts continuously past the freezing-point. But because no final assumption is necessary, we may leave the question open and assume provisionally that the contraction is continuous before, during and after freezing, and see why it is under this assumption that the pipe forms. Next, passing from the simple to the complex, let us see how it may come about that a pipe yet

[5] *Comptes rendus*, vol. cxl., pp. 185 to 192 (1905). See § 18, p. 31, of the present paper.

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may form in a substance which, like gray cast-iron and water, certainly expands near the freezing-point; and if it should later be proved that steel expands in freezing, the explanation given for the piping of ice and gray cast-iron would suffice to explain the piping in steel.

But the change of volume of steel in freezing, whether it is an expansion or a contraction, is probably small, not comparable with the expansion of ice and gray cast-iron, and wholly incompetent to explain the often large volume of a pipe.

In passing, let me explain why this evidence that steel contracts in freezing seems to me inconclusive.

That scrap-steel is not seen to float in certain basic open-hearth practice is inconclusive for several reasons. If it projected but slightly above the surface of the molten bath, this projecting might well be so masked by the overlying thick layer of slag as to pass unnoticed. Again, if the scrap does sink, this may be because the bath into which it is charged is so much richer in carbon as to be materially lighter than the scrap-steel itself would be if molten. The question is not whether low-carbon steel scrap will float on molten high-carbon steel or on molten cast-iron, but whether it will float on molten steel of its own composition. Finally, even if steel does expand in solidifying, yet cold steel scrap might still sink in molten steel of its own composition, for even gray cast-iron when cold sinks in like cast-iron when molten, and only after it has grown hot does it again rise to the surface. In case of steel scrap charged cold, it might not as a whole be heated near enough to the melting-point to become light enough to float, until so much of it had melted away that the little which remained would project so little as to be masked by the overlying slag.

The sinking of solid steel bars in the casting-ladle in my own experiments is not conclusive, because the molten steel was very far above its melting-point and the scrap was very far below its melting-point.

Moissan's results are inconclusive for reasons which we will take up in § 18 after further study of this general question, and the high position of the segregate is inconclusive for reasons explained in § 46, p. 68.

On the other hand, the very common observation, my own included, that pre-heated solid scrap-steel floats in the acid

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open-hearth furnace is not strictly conclusive, because this scrap is charged through the molten slag, and enough slag may adhere to it to buoy it up. Again, gas-bubbles may buoy it up, quite as the bubbles in a glass of champagne will buoy up crumbs of bread dropped in. I attach little weight to this explanation, because the buoying action of these bubbles ought to be very irregular, making the scrap bob up and down, with a strong evolution of gas around it, whereas my observation has been that the scrap floats quietly if the bath itself is quiet.

Again, the pipe in steel ingots (except in very narrow ones, as explained in § 38), instead of being open as it is in the zinc ingots of Fig. 15, is covered with a level crust, often of considerable thickness. From this we might at first infer that the metal expands in freezing ; but, as I will explain in § 14, such an inference would be unjustified.

§ 6. *Another explanation offered.* Carrying our analysis further, let us divide the freezing-period into two parts, one which precedes and one which accompanies the formation of the pipe, or into the "pipeless" and the "piping" periods; and let us divide the shell of the ingot into two imaginary concentric layers, an outer very thin one, as thin as you please, and an inner one which comprises all the rest of the metal which solidifies during the ante-piping period, or which has solidified up to any special time under consideration. These two parts we may, for brevity, call the outer walls and the inner walls. Note these meanings.

This done, my explanation is that during the pipeless period the outer walls are virtually and permanently much expanded by the resistance which the inner walls, and the molten metal too in case the top of the ingot is firmly frozen over, oppose to their normal contraction; and that during the piping period the contraction of the inner walls, welded as they are to the outer walls thus virtually expanded, causes them to draw outwards, leaving an empty space, the pipe.

It is not only that the volume of the molten interior, at the moment of solidification of the outer crust, determines the initial dimensions of this crust, but more especially that the lagging cooling of the inner walls determines afterwards the virtual expansion of this crust; and it is rather the contraction of the inner walls after freezing, than the contraction of the

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molten interior before and during freezing, that later makes the interior fall short of filling the outer shell, and thus causes the pipe.

Let us now go on to consider the pipeless and the piping periods separately.

§ 7. *Virtual expansion of the outer walls in the pipeless period.* When freezing first begins, the very outermost layers—that is to say, the outer walls—cool much faster than the layers within them—that is to say, the inner walls—because the former are in immediate contact with the cold walls of the mold. Cooling faster, they tend to contract faster; but this tendency to excess of contraction is resisted by the inner walls. The outer walls are like a tire shrunk upon a wheel which itself is contracting more slowly than that tire. At the very first this resistance of the softer and weaker inner walls may have little effect; but as they grow thicker and thicker, and as their outer part becomes firmer and firmer, so do they resist the more effectually the endeavor of the outer walls to contract faster than the inner ones. In so far as this resistance is effectual, it virtually expands the outer walls, in the sense that it prevents them from contracting at the normal rate which they would have followed but for this resistance, so that at each temperature they are larger than they would have been had their contraction been unresisted. By as much as they are thus larger, by so much are they virtually expanded.

§ 8. *Virtual expansion persists.* By as much as the virtual expansion of the outer walls, which has taken place up to the time of reaching any given temperature in the cooling, has exceeded the then existing elastic limit, by so much will it tend to persist during the remainder of the cooling. If, for instance, on reaching a temperature of 1,000° C. the virtual expansion were such that the ingot was 1 in. wider than it would have been if the contraction of the outer walls from their initial size had been unresisted, and if by some device the inner walls could be bodily removed from within the outer walls, these latter, resuming their natural rate of contraction, would contract from their present size as a datum size; and at the end of the cooling they would remain 1 in. wider than they would have been had their contraction been unresisted from the very beginning.

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Of course, the virtual expansion which has taken place up to any given moment may be augmented later by the continued working of like causes; or it may be lessened by centripetal stress either from without (such as atmospheric pressure) or from within. For our present purpose it suffices that any virtual expansion tends to persist.

§ 9. *Later the inner walls contract faster than the outer walls.* Although at the beginning of the freezing the outer walls, because of their contact with the initially cold walls of the mold, cool faster than the inner walls, later a time must come when this is reversed and the inner walls cool faster than the outer ones, as the least reflection makes clear. Suppose, for instance, that at some particular instant the outer walls had cooled from 1,600° C. to 600°, while the average temperature of the inner walls was 1,100°. In the remainder of the undisturbed cooling, say to 20°, the temperature of the surrounding air, the outside has to cool through only $600 - 20 = 580°$, while the inner walls have, as a whole, to cool through $1{,}100 - 20 = 1{,}080°$, or nearly twice as far. Outer and inner walls will arrive at 20° at practically the same time; for so long as the inner walls are above 20° by any appreciable amount, the outer walls will also be slightly above 20°, because through those outer walls must pass the heat which is escaping from within, and that heat cannot in turn pass from those outer walls into the surrounding atmosphere, which is at 20°, unless those outer walls are themselves above 20°; for in effect heat, like water, will flow only from a higher to a lower level. Thus the inner walls, as a whole, have to cool through 1,080° while the outer walls are cooling only through 580°; this journey the outer and inner walls must make in practically the same time; so that clearly through at least part of their longer journey the inner walls must travel faster than the outer ones.

Consider the case of a horizontal tube, Fig. 3, full of water, which is held in by a tight cork. Withdraw the cork and the level of the water in the very mouth of the tube at first falls very fast, much faster than the level within the tube; but soon the level in the mouth grows nearly stationary, while the level within falls progressively, and falls faster than the level in the mouth.

§ 10. *The piping period.* And this brings us by an easy stage

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to the piping period. As the amount by which the cooling and the consequent tendency to contraction of the outer walls outrun those of the inner walls progressively lessens, first the resistance which the lagging contraction of the inner walls offers to the faster contraction of the outer walls grows less and less, so that the virtual expansion of the outer walls becomes slower and slower, and the outward pressure of the inner walls against them lessens correspondingly. Soon the rate of cooling of the inner walls catches up with that of the outer ones, so that the former cease to press outwards against the latter, and to increase their virtual expansion; the outward pressure of the inner against the outer walls falls to zero; and the virtual expansion of the outer walls ceases to increase. The outer walls now have "got their growth," as we say of a young man; and though henceforth their actual dimensions continue to shrink, this no more effaces the virtual expansion which they have now undergone than the natural shrinkage of the giant's stature in old age makes him cease to tower above his pigmy classmates, whom age is shortening with like cruelty.

FIG. 3.—RATE OF FALL OF THE UPPER SURFACE OF WATER IN DIFFERENT PARTS OF A TUBE SUDDENLY UNSTOPPERED.

The fall of the upper surface is at first most rapid in the mouth of the tube, but later most rapid in the interior.

And next, as the rate of cooling and contraction of the inner walls outruns that of the outer ones still further, the volume of the inner walls plus that of the still molten lake ceases to fill completely the outer walls. And now the pipe begins to form: for the excess of contraction of the inner over that of the outer walls makes these inner walls draw outwards, as explained in the next section, and so enlarges the cavity between the inner walls in which lies the submerged molten lake. This lake, therefore, ceases to fill its cavity completely, so that its upper surface begins to descend, its tide to ebb.

§ 11. *The contraction of the inner walls enlarges the central cavity.* That this contraction of the walls of a cavity may result in the expansion of the cavity itself, is clear when we consider a case like that sketched in Fig. 4, a block of soft india rubber with a cavity which normally has the shape *abcd*, shown in dotted lines. By means of the strings, *he*, *kf* and *lg*, we stretch

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the rubber inwards and lessen the cavity by deforming it into the shape shown by the full line, *aefgd*. If these strings are now released, the rubber which their pull has expanded will contract back to its original shape, but in thus contracting the walls of the cavity draw apart from the shape *aefgd* to the shape *abcd*, and the cavity itself thus expands.

Or consider the case of a bottle with its outer walls of glass, which, though itself expanding and contracting with changes of temperature, is at each temperature rigid and unyielding. Consider further that to the inner walls of the glass is firmly welded a thick lining of soft india rubber, which reduces by

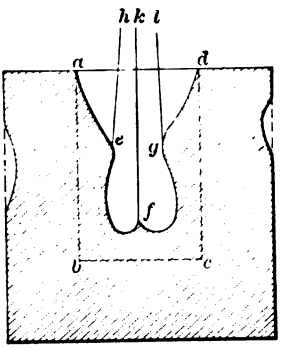

FIG. 4.—THE CONTRACTION OF THE WALLS OF A CAVITY ENLARGES THE CAVITY ITSELF.

The inner walls of the cavity, *abcd*, in a hollow block of soft india rubber are here supposed to have been drawn inwards by means of the strings *he*, *kf*, and *lg*. On releasing those strings, the contraction of the india rubber in returning to its original shape enlarges the cavity to its original size.

perhaps one-third the holding-capacity of the bottle. If the glass shell expands, it draws the rubber after it and enlarges the cavity; if it contracts, it pushes the rubber inwards and lessens the cavity. But if, while the glass shell remains constant in volume, the rubber lining expands, since the rigid glass walls prevent it from moving outwards, its expansion takes place inwards, lessening the cavity; while if it contracts, since it cannot draw the glass walls inwards, its contraction makes it draw out towards those walls, and thus enlarges the cavity.

To sum this up:

If the { outer / inner } walls expand they { enlarge / lessen } the cavity;

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If the { outer / inner } walls contract they { lessen / enlarge } the cavity.

During the pipe-forming period the rate of contraction and the consequent tendency to outward movement of the inner walls are outrunning the rate of contraction and the consequent tendency to inward movement of the outer walls. This causes the opposite shores of the cavity which contains the molten steel to draw apart outwards, and thus enlarges the cavity between these shores, so that the molten lake ceases to fill this cavity completely, and its upper surface, instead of pressing against the top of the cavity in which it lies, begins to sink down from that top, its tide begins to ebb, and thus begins the pipe, as sketched in Fig. 1.

As this state of affairs continues, so does the pipe continue to form. The sum of the contraction of the inner walls plus that of the molten lake continues to outrun the contraction of the outer walls; the inner walls plus the molten lake fall further and further short of filling the outer walls, and therefore the pipe or cavity, which represents this deficit of volume, continues to grow, and the level of the molten lake continues to fall, narrowing all the time, as layer after layer of the lake freezes against its own shores.

§ 12. *Summary.* We have now seen in § 7 how the resistance of the initially slower-cooling inner walls virtually expands the faster-cooling outer ones; in § 8 that this virtual expansion is persistent; in § 9 that later the inner walls in turn cool faster than the outer ones; in §§ 10 and 11 that the excess of contraction of the inner walls in this later period, welded as they are to the outer ones, enlarges the cavity between those walls, so that the molten lake between them, which itself is contracting, ceases to fill this cavity fully, its tide ebbs, its surface falls, and this ebb causes the pipe.

I advise most readers to skip §§ 13 to 16 A, inclusive, pp. 17 to 30, because the discussion is very technical and the reasoning very close.

§ 13. *The effect of expansion at or near the freezing-point on the volume of the pipe.* In reaching our conclusion that (1) the virtual expansion of the outer walls, due to their fact that their cooling at first outruns that of the inner walls, joined to (2) the

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excess of contraction of the inner walls over that of the outer walls in the latter part of the solidification, gives rise to a pipe, we assumed for simplicity that the contraction was continuous and uniform in passing the freezing-point, as indicated by the line *GH* in Fig. 5. Let us next ask how the volume of this pipe should be affected by interrupting this uniform contraction, and replacing it for a time with expansion. To most readers it will at first seem self-evident that any expansion at or near the freezing-point must lessen the pipe, and not a few will impatiently brush aside any objections to this inference. But a closer examination shows that the matter is not to be disposed of so lightly. In particular we must not forget that

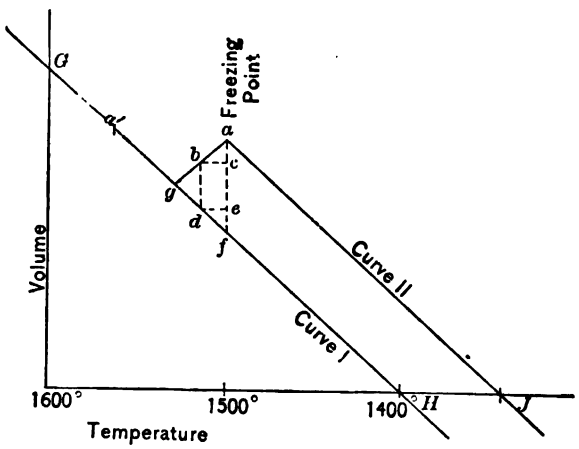

FIG. 5.—EXPANSION ABOVE THE FREEZING-POINT.

such an expansion, though when it is occurring in the freezing of the axial metal it certainly tends to lessen the pipe, must also have previously occurred in the freezing of the outer shell at the time when it froze, and must thus have tended to increase the pipe; because after all the pipe represents simply the difference in volume between the outer shell of the ingot and the interior which is the contents of that shell. The pipe is simply the deficit by which the volume of the interior falls short of filling the shell completely, at the moment when the last of the axial metal has become too solid and strong to be further opened or shut by the interstratal movements of the ingot.

To clarify our ideas, let us consider the case of a nest of

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hollow concentric brass spheres, each fitting exactly over the one next inside it. No matter how such a nest of spheres is to be made; leave that to me, and simply picture such a nest in your mind. If heat is applied from outside, the outer sphere expands and gapes slightly away from those within it; but if, for instance, the whole nest is immersed in boiling-water and left there till the very center has reached 100° C., then in effect the heating will have sent a gradual wave of expansion from the surface of the outer sphere radially inwards to the center of the inner sphere. This expansion will have expanded first the outer sphere, and in turn each successive inner one, and finally the central one, until at the end, when all have reached 100°, all will have expanded exactly alike, and each will fit the next one inside it, its neighbor, quite as exactly as it did in the beginning. The wave of expansion will have expanded all the members of the nest, not all simultaneously, but in due time all proportionally. Had there been a hollow or pipe in the inner sphere, and had this sphere alone expanded, and had the outer ones, not expanding, forcibly resisted this expansion and prevented it from taking effect outwards, then it would have pressed inwards, and this would have tended to lessen that pipe; but a wave of expansion which gradually traveled inwards, expanding each sphere in turn proportionally, would not thus tend in the least to close any such initial pipe in the central sphere.

This conception or its equivalent is almost necessary to a clear understanding of the subject, and the reader should not pass on till his mind admits the truth of this picture as absolutely unquestionable.

Those who say glibly that expansion at or near the freezing-point must tend to close the pipe, forget that this same expansion may already have increased the pipe-forming tendency by an equivalent amount, when it correspondingly expanded the freezing outer crust of the ingot. We must therefore look deeper, and see in what indirect way this expansion may affect the volume of the pipe. In particular, let us note that a freezing ingot differs from our imaginary nest of brass shells in having cohesion between each layer and the adjoining layers. Therefore, let us ask in particular how this interstratal cohesion may affect the result. We shall find that careful

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study is needed to permit us to draw any inferences, and those of us who are cautious will even then take these inferences with some reservation.

Let us divide our question into three parts, asking first the effect of an expansion which takes place wholly above the freezing-point; next, the effect of one which takes place wholly at the freezing-point; and, finally, the effect of one which takes place wholly below the freezing-point, as sketched in Figs. 5, 7 and 9, respectively (pp. 18, 25, and 27).

§ 14. *Case* 1, *the expansion takes place wholly above the freezing-point.* Here our question is, "How is the volume of the pipe affected by a change in the volume temperature-curve, from the line *GgH* to the line *GgaJ* of Fig. 5?"

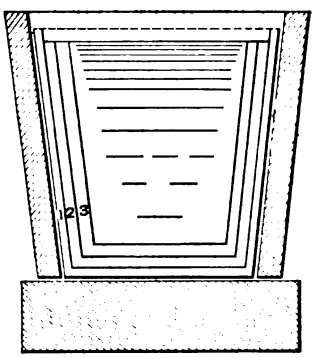

FIG. 6.—THE FREEZING OF A MASS CONSIDERED AS A NEST OF CONCENTRIC SHELLS.

Let us consider the molten metal as made up of a series of concentric shells, Fig. 6, assuming for simplicity that each of these is at uniform temperature throughout.

Sub-case A. As the metal begins to freeze, it is by diligent stirring brought to exactly uniform temperature throughout, so that it is represented by the point *a* in Fig. 5. Each layer, as it now freezes and cools, will travel down the line *aJ*. But, as this line is exactly parallel with *gH*, the line which would have been followed if the expansion had been continuous and uniform, each layer in thus cooling through the same number of degrees will undergo the same amount of contraction in both cases, so that substituting *GgaJ* for *GgH* really has no influence on the contraction at all, and therefore none on the size of the pipe. In other words, because by assumption the whole of this expansion in cooling has taken place before the freezing

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actually begins, and because it is only the conditions which exist during freezing that lead to formation of the pipe and affect its size, this expansion can have no effect on the pipe. If the line *GgH* would lead to a pipe, the volume of which would be 3 per cent. of that of the ingot itself, the line *GgaJ*, if in effect it thus begins only at *a*, also will give a pipe of which the volume will be 3 per cent. of that of the ingot. In short, the relative volume of the pipe is unchanged; but, as the expansion, *ga*, increases the absolute volume of the ingot, so will it increase in like ratio the volume of the pipe.

Sub-case B. When the outer shell (1 of Fig. 6) is beginning to freeze—*i.e.*, is at temperature *a* of Fig. 5—the interior is at temperatures between *a* and *g*, and its average temperature is *b*. One obvious effect of this supposed increase of volume before reaching the freezing-point, is to make layer 1 lighter than the rest, so that it swims to the upper surface. But to simplify our study, let us assume that the heat is removed so rapidly from the outside that this upward swimming of the cooler and lighter parts has not time to take effect to any important extent, so that each of the concentric shells, 1, 2, 3, etc., of Fig. 6, remains in place during the cooling and freezing.

When the infinitely thin shell 1 reaches temperature *a* with curve II. it is larger by the distance *af* than it would have been with curve I. But the dimensions of this still liquid shell 1 are determined, not by the expansion which it has now undergone, but by that which the average of the metal within it has undergone. If, for the moment, we conceive that, following curve II., this shell 1, expanding more than the metal within it, raised its edges above the level of that metal, as sketched in dotted lines in Fig. 6, we see that the metal which was thus raised above the general level would at once sink to that level, of course raising that level proportionally. By like reasoning, if curve I. is followed, the volume of the outer shell, when it had fallen to temperature *f*, would be determined by the volume *d* of the interior at its average temperature, assumed as *d*, temperature *b* and temperature *d*, of course, being identical. At this moment, then, the outer shell becomes solid, or freezes; and this first solid outer shell is just large enough to contain the still molten interior.

In the further cooling, the difference between the behavior

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with these two curves is that with curve I. the natural contraction of the interior in further cooling from *d* to *f* exceeds that of shell 1 by *ef*, whereas with curve II. it falls short of the natural contraction of the shell 1 by *ca*. Now, if the natural contraction of the shell could take place unobstructed, the effect of this difference ought to be to diminish the pipe by an amount corresponding to the sum of *ef* plus *ca*. In other words, with curve II. the outer shell 1 is naturally larger by *ce* than it is with curve I., because the interior is thus larger at the moment when freezing determines the initial volume of the shell; but the interior, when it in turn reaches the freezing-point, will be larger by *af* with curve II. than with curve I.; so that substituting curve II. for curve I. implies increasing the volume of the interior, as it reaches the freezing-point, relatively to shell 1, by $af - ce$; and the fact that this is equal to $ef + ac$ confirms the inference reached in the preceding sentence.

The volume of the pipe, as we have already seen, is the difference between the volume of the outer inclosing shell and the volume of the contents of that shell at the time when the contents is just reaching that degree of rigidity which prevents further change of the volume of the pipe. In this particular case when, in freezing, the dimensions of the outer shell are determined by that of the still-liquid interior, the material which composes that shell has already done its expanding, which takes place wholly above the freezing-point. Therefore, if the molten interior could now be removed from within this first formed shell, the shell itself would henceforth contract in following curve II. exactly as it would have done in following curve I. From now on the substitution of curve II. for curve I. has no effect on the contraction which the outer shell would follow if unobstructed. The outer crust travels down *aJ*, quite as if it were traveling down *gH*.

But, while the material which composes the outer shell has now already done its expanding, that which composes the interior has not. From this time on, the substitution of curve II. for curve I. means that the net contraction of the interior is lessened by $ef + ac$. This lessening of the contraction of the interior should lessen the volume of the pipe by $ef + ac$, and the lessening should persist through any and all subsequent changes of volume.

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That this last assertion is true is seen by comparing the cases of two trees, growing at very different rates, with different and complex variations of rate in different seasons. If the pot in which one tree is standing is arbitrarily lowered 1 ft. below that at which the other stands, this initial difference of 1 ft. will change all the subsequent differences of height between these two trees by exactly 1 ft., no matter how complex may be the variations of growth of either tree.

The foregoing reasoning is based on the supposition that the natural contraction of the outer crust is unobstructed. Yet it clearly must be obstructed by the expansion of the metal within it, which must tend to stretch it. If this stretching did not exceed the existing elastic limit of the outer shell, the obstruction would be only temporary, and at the end the pipe would actually be lessened by the amount $ef + ac$. But most metals have so low an elastic limit when they have first frozen, that much of the stretching of the outer crust by the expansion of the interior must result in a permanent set, and thus be permanently effective.

In the early part of the freezing, even of a material which contracts continuously and uniformly past the freezing-point, the more rapid cooling of the crust than of the inner walls brings about, as we have seen (§ 7), a virtual expansion of those outer walls. I have likened this to shrinking a tire upon a wheel which is contracting, but contracting more slowly than the tire itself. But in our present case the expansion which the interior is undergoing as it approaches the freezing-point aggravates the matter, which must be likened to shrinking a tire upon a wheel which itself is expanding.

Thus, in the early part of the freezing, much of the relative expansion, $ef + ac$, of the interior relatively to the outer crust expends itself in increasing the expansion of the crust, perhaps even turning a virtual into an actual expansion. It is only in the latter part of the solidification, when the rate of cooling of the interior has not only caught up with that of the outer shell, but has so far outstripped it that the average rate of contraction of the inner walls exceeds that of the outer walls by an amount greater than the simultaneous rate of expansion of the still molten interior—it is only then that the excess, $ef + ac$, of the contraction of the interior will begin to take effect in

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lessening the volume of the pipe. But some remainder of this expansion will do this, quite as a stream cannot rise above its source. It is to be remembered that, as long as the expansion of the molten interior is resisting the contraction of the crust, so long is the beginning of the formation of the pipe postponed, because no vacant space can arise while the molten interior is still pressing against the contracting shell.

Here is another way of looking at it, which leads to the same conclusion. Substituting curve II. for curve I. causes an expansion of the interior relatively to the crust by $ef + ac$. This will in turn lead to an expansion of the crust, not by the whole, but by part only, of $ef + ac$; and the remainder of $ef + ac$ is therefore an effective lessening of the pipe.

This general result is the same whether we imagine that the freezing takes place not only at the bottom and sides of the ingot, but also equally rapidly at its top; or whether we imagine that the top does not freeze over till the very end of the freezing. In the latter case the expansion of the interior, instead of increasing the virtual expansion of the outer walls in every direction, simply raises the level of the upper surface; but this, too, must eventually freeze over, and when it does freeze its previous rise will have increased the total volume of the ingot quite as much as in the other case.

In short, of the excess, $ef + ac$, of the expansion of the interior, part will be consumed in increasing the volume of the shell, and only the remainder will take effect in lessening the volume of the pipe.

To sum this up, if (sub-case A) the metal is all brought to the freezing-point at the same instant, expansion before freezing can have no effect on the volume of the pipe. If (sub-case B) at the moment when the outer crust freezes the interior is somewhat above the freezing-point, then part of its expansion in approaching the freezing-point will result in expanding the crust itself, but still part of its expansion will take effect in lessening the pipe.

§ 15. *Case* 2, *the expansion takes place wholly at the freezing-point*, at the instant when the metal is passing from the liquid to the solid state, substituting curve III., Fig. 7, for curve I. This introduces a wholly new and different condition: that the layer which at any instant is expanding by the distance ga is

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now solid metal, tending to retain its own shape and dimensions, instead of being liquid, as in case 1, and flowing to conform with whatever mold surrounds it. Moreover, each layer in freezing and expanding may be assumed to cohere strongly

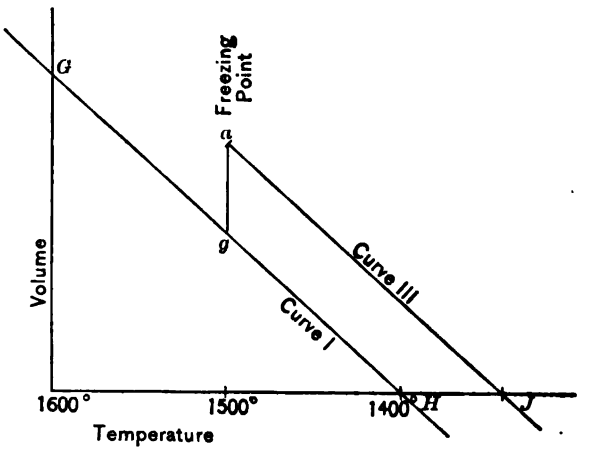

FIG. 7.—EXPANSION AT THE FREEZING-POINT.

with the solid layer outside it, but not to cohere strongly with the still molten layer inside it.

Were it not for this second consideration, the case would seem to resemble that of our nest of hollow brass spheres, one within another, considered early in § 18. But in case 2 the cohesion of the layer which is freezing and expanding with

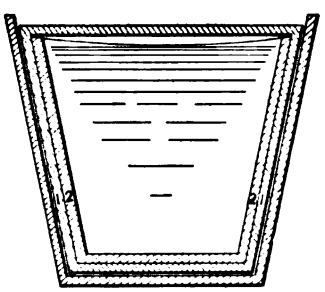

FIG. 8.—THE FREEZING OF A MASS WHICH EXPANDS AT THE FREEZING-POINT.

the layer outside it, which has already frozen and finished its expansion, may be expected to add to the expansion of that outer layer. Thus, in Fig. 8, the outermost layer of all, 1, when it starts to freeze has the dimensions of the still-molten mass within; but in the act of freezing it expands. Each of a

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series of infinitely thin layers will be freezing and expanding only an infinitely short time, so that the expansion of each will be finished before that of the next begins. But these layers, though infinitely thin, are of very considerable length, nearly the length of the ingot itself, so that the amount by which each expands is considerable. Layer 1 in freezing, therefore, raises itself up above the surface of the still-molten mass within it, so that a narrow empty space tends to form between the top of the molten metal inside and the crust of shell 1, which now incloses it. Layer 2 in freezing in turn tries to expand, but its expansion is interfered with by its cohesion with layer 1. The result of such an interference is a compromise; the expansion of layer 2 takes place, but not to its full extent: in so far as it takes place it expands layer 1, which thus remains in tensile stress, and in so far as the expansion is suppressed layer 2 is shortened and prevented from reaching its natural growth, or is virtually compressed, and remains in compressive stress. Next comes layer 3, which in turn tries to expand, and acting through layer 2 throws additional tensile stress on layer 1, and expands it further; and so on with each successive layer. In so far as this action expands the outer crust, it tends to increase the pipe, like any other form of virtual expansion of the crust.

And this, it seems to me, is the natural result of an expansion which is strictly limited to the instant of solidification. It is true that, as solidification proceeds, the influence of the expansion of each successive freezing-layer in virtually expanding the outer shell is less and less, because that outer shell is always growing colder and hence stronger; so that a larger and larger proportion of the natural expansion of each freezing layer takes effect in making that layer bulge inwards and thus lessen the pipe. But even if the expansion of each and every layer, including the outer one, was its full natural expansion, we should reach only the condition of our nest of progressively heating and expanding solid brass shells, which, when all have heated and expanded alike, just fill and fit one another exactly as they did before any expansion began. If there were no cohesion between the successively freezing layers, the effect of a wave of expansion which traveled inwards from layer to layer and ended at the central last-freezing point

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just as that point froze would be *nil*, for it would expand each successive layer just proportionally to every other layer. Therefore if, as I suppose, the cohesion of each freezing layer has a certain effect in expanding the layers outside it, and this in turn leads to some expansion of the outer shell in excess of the normal, then, however slight this excess, it should add itself to the volume of the pipe, because the expansion of the interior cannot exceed the normal, and thus cannot make up for the excess of expansion of the crust.

§ 16. *Case* 3, *the expansion takes place wholly after solidification*, so as to substitute curve IV. of Fig. 9 for curve I., but yet not very long after solidification, so that at the time when the

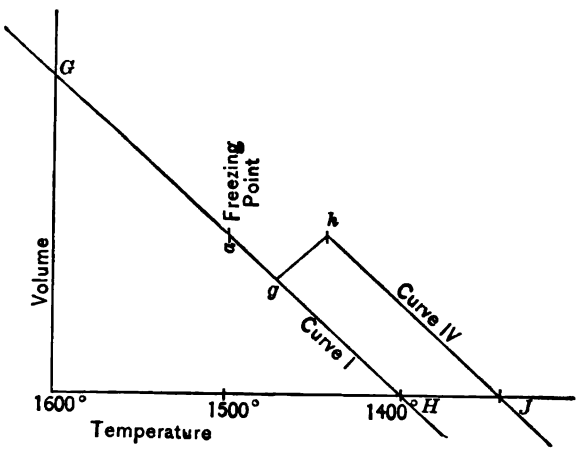

FIG. 9.—EXPANSION BELOW THE FREEZING-POINT.

outer shell is thus expanding from g to h, much of the interior is still unfrozen.

Here first a wave of solidification or freezing sweeps gradually through the ingot, from its outer shell inwards to the last-freezing point; and, after this first wave has started to sweep inwards, but before it has reached the axis of the ingot, a second wave, an expansion, follows the first.

If the ingot were composed of a nest of wholly unconnected concentric shells, this wave of expansion would have no effect on the relative volume of pipe and ingot. Each shell would expand in exactly the same proportion as every other shell. If the expansion were, for instance, 3 per cent., the absolute volume of ingot and of pipe would expand by 3 per cent., and the ratio

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of pipe to ingot would be unaffected. The question before us then is, "How does the cohesion which exists between the different layers affect this result?"

As the outer shell starts to expand from *g* towards *h*, its cohesion with the cooler layers within it, which are still contracting, because they are still moving in the direction *ag*, impedes its expansion and throws it into compressive stress, while by this same fact the layers within it are thrown into tensile stress, and slightly expanded beyond their natural dimensions. When layer 1 begins to contract from *h* towards *J*, layer 2 within it is traveling up from *g* towards *h* and expanding, and its expansion is in turn resisted by the contraction of layer 1, and by that of layer 3, which is traveling in the direction *ag*, and so on. Thus the tendency of each successive layer to expand is impeded by its cohesion with the neighboring ones which are simultaneously tending to contract, so that this wave of expansion is (1) immediately preceded by a wave of tensile stress, is (2) accompanied by a wave of compressive stress, and is (3) immediately succeeded by a second wave of tensile stress.

If these several stresses did not reach the existing elastic limit of the material as the wave gradually swept inwards, so that layer after layer reached *h*, and in turn traveled thence towards *J*, these stresses would eventually efface each other when the innermost layer of all had reached *h*, because every layer would have undergone the same expansion. Therefore, our next question is, "If the stress is in excess of the elastic limit, what relation will this excess in the outer layers bear to the excess in the central layers?" This we ask, because it is the difference between the volume of the outer shell and that of the interior that determines the volume of the pipe.

Now there is one extremely important difference between the conditions of the outside and those of the deep-seated interior, at the respective times when each is expanding from *g* to *h;* and that difference is that the wave of expansion travels much more rapidly in through the quickly cooling shell of the ingot than through the relatively-slowly cooling deep-seated parts. On this account, for a given distance between two neighboring layers, the temperature-difference will be much greater in the outer layers at the time when the wave of expansion is passing there, than in the deeper-seated layers when the wave

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in turn reaches them; and the greater the temperature-difference, the greater will be the stress set up by the wave of expansion, the greater also will be the amount by which that stress exceeds the elastic limit of the material, and hence the greater will be the permanent set, whether in tension or in compression, which is set up. Therefore, the permanent set, if there is any, will be greater in the outside layers than in the deep-seated ones.

Will this permanent set be a compression or an extension? What concerns us chiefly is the permanent set in the very outside layer, and this, it seems to me, must be a compression, because at the moment when the very outside layer is expanding, its expansion is resisted by the tendency of the whole of the material within it to contract. Indeed, unless the expansion which we are imagining, *gh*, is a most extraordinarily great one, it seems to me that the permanent set in compression should exceed that in extension throughout the whole travel of the wave from outside to axis, because the number of layers which at any given moment are trying to expand is much smaller than the number of those which at that moment are trying to contract; in other words, the resistance of the temporary majority of the layers, the party of contraction, should outweigh that of the minority, the party of expansion; and, though each may whittle down the influence of the other, the expanding tendency of the layers which are expanding causing a slight permanent extension in those which are trying to contract, and *vice versa*, yet in each of these struggles the greater whittling down will be done by the layers which are temporarily in the majority, and this will always be the party of contraction.

The sum of this is that greater permanent set is to be expected in the outer shell than in the interior, and that this set should be chiefly in compression; or, in short, that the effect of an expansion after solidification should be to lessen the volume of the outside shell rather more than that of the interior, and this evidently should tend to lessen the volume of the pipe.

§ 16, A. *Case* 4, *the expansion takes place in the freezing-range, as distinguished from the freezing-point.* Here I refer to the fact that the freezing of carburized iron, like that of most alloys, takes place not at a single point but through a very consider-

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able range of temperature, which, in case of iron containing 2 per cent. of carbon, reaches from 1,130° to 1,325°. In this case, at any given instant during freezing, there is a series of concentric shells, all of which are in this freezing-range, and all of which, therefore, are freezing. The layer which at this instant forms the boundary of the molten lake is just beginning to freeze; and as we pass outwards the freezing of each successive layer has, at this same instant, gone somewhat further than that of the layers within it.

Of these several freezing layers, only the innermost, or a few of the innermost ones, truly come under our case 2, the essence of which is that the freezing and expanding layer does not cohere with those within it, though it does cohere with those outside it. All but a few of the innermost of the layers, which at any given instant are freezing, fall under case 3, in which the expanding layer coheres both with those outside and with those inside it.

If freezing covers any considerable range of temperature, then the number of layers which thus fall under case 3 should be far greater than the number of those which belong under case 2; so that the net effect of the expansion, like that of case 3, should be to lessen the pipe.

The freezing of gray cast-iron probably belongs to this case 4. The floating of the graphite, set free during solidification, tends to show that there is expansion at the very moment when freezing begins, because the graphite, if it were not set free until freezing had progressed far, or until it had ended, would be mechanically prevented from rising to the surface; and the formation of so light and bulky a substance as graphite in the freezing metal must make it expand. Beyond this, the general experience of the annealing process for making malleable cast-iron, like the behavior of black-heart file-steel, tends to show that the formation of graphite continues at temperatures considerably below the end of the freezing-range proper.

§ 17. *General summary of the influence of expansion at or near the freezing-point.* To sum up the foregoing discussion:

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If the expansion takes place wholly	The effect on the pipe will be
Case 1, A. Above the freezing-point, and the whole is brought to uniform temperature before freezing begins	*Nil.*
Case 1, B. Above the freezing-point, and the whole is not brought to uniform temperature before freezing begins	To lessen it.
Case 2. At the freezing-point	To enlarge it.
Case 3. Below the freezing-point	To lessen it.
Case 4. Through the freezing-range	To lessen it.

Thus expansion tends to lessen the pipe in all except the rare cases in which it takes place either wholly above the freezing-point in a liquid brought throughout to constant temperature, or at the true freezing-point of a pure substance, which really freezes at a single point instead of through a range of temperature.

§ 18. *Moissan's evidence.* The reasoning which we have been following puts us in a position to consider Moissan's evidence, which led him to infer that, although cast-iron containing from 7.65 to 8.17 per cent. of carbon does expand in solidifying, yet steel containing less than 1 per cent. of carbon does not, but follows the general law and contracts. His evidence is in effect that, whereas a button of such cast-iron in solidifying in the air bursts its shell and spurts out a considerable mass of molten metal, yet a little button of steel, made by melting 500 g. of pure Swedish wrought-iron in a magnesia crucible, does not thus burst its shell, but becomes hollow (*se creuse*) and solidifies without the escape of anything from its interior. The expulsion of the molten cast-iron from the solidifying cast-iron button he properly refers to the expansion of the freezing interior, which, taken jointly with the evolution of gas in freezing, and the fact that the cooling and contraction of the outer crust outrun those of the interior, suffices to burst open the contracting outer crust.

But the phenomena can readily be explained on the hypothesis that steel does expand in solidifying, especially if we concede that its expansion ought to be less than that of cast-iron, the expansion of which should be increased by the formation of graphite within it.

Thus, first, he found that like buttons of gray cast-iron, if

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cooled suddenly, did not always eject anything from within; which may be taken to mean that, even with the greater expansion of this cast-iron in freezing and with the weakness and brittleness of the inclosing envelope itself (that is to say, the outer crust of already solid cast-iron), yet under favoring circumstances this weak and brittle cast-iron crust suffices to resist so completely the expansion of the freezing metal within that this crust is not broken through, and none of the molten iron from within escapes. If this is possible under favoring conditions with so brittle a crust, and with such great expansion as we have in case of cast-iron, it is not at all surprising that the far stronger and more ductile outer steel shell of a solidifying steel button should suffice, under less favorable circumstances, to resist unbroken the much slighter expansion of the freezing steel within it.

The reasoning which we have studied suffices to show that a pipe or other hollow space may well form even in materials which expand in freezing. The presence of a pipe in very graphitic cast-iron is no rarity; so that the fact that the metal becomes hollow (*se creuse*) is no proof that it does not expand in solidifying.

§ 19. *The crust which covers the top of the pipe in steel ingots.* The fact noted in § 5, that the pipe in steel ingots, except in very narrow ones (§ 38), instead of being open at the top like those in Fig. 15, is covered by a crust, often of considerable thickness, at first suggests that the metal expands in freezing. It certainly does imply that the upper surface of the molten steel, instead of beginning to sink down as soon as the outer walls of the ingot begin to solidify, remains level long enough to freeze across. But this I refer, not to the expansion of the steel itself in solidifying, but to the fact that the more rapid cooling of the outer than of the inner walls at the beginning of freezing forces these inner walls inwards, and so raises the level of the molten metal, or at least prevents it from sinking. The failure of the upper surface to sink at once is probably due, not to the expansion of the molten metal itself, but to the squeezing inwards of the inner walls of the ingot by the now more rapid contraction of the very outer crust.

§ 20. *Can a substance which expands at or near the freezing-point yet form a pipe?* It clearly can. All that is necessary is

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that the pipe-forming tendency, due to the virtual expansion of the outer crust, shall exceed the pipe-closing tendency, due to the expansion near the freezing-point. But, though this expansion, according to our reasoning in §§ 13 to 16 A, may lessen the pipe, as is illustrated by the fact that gray cast-iron, which expands apparently shortly below the freezing-point, pipes less than white cast-iron, yet whether it shall efface the pipe or not depends on the strength of the pipe-forming tendencies.

§ 21. *What arrests the down-stretching of the pipe?* At first the reasoning in § 11, p. 15, seems to prove too much, implying that the pipe ought to reach nearly to the very bottom of the ingot, whereas, in fact, it very rarely reaches down one-third of the way. Thus, considering any given horizontal layer, for instance, layer V. in ingot *E* of Fig. 1, by our reasoning when the axial part, 3, reaches its freezing-point, the inner walls 2 and 4 should still be cooling faster than the outer walls 1 and 5; and by as much as their contraction at this time is outrunning the contraction of 1 and 5, by so much should this layer, as a whole, be drawing outwards, and tending to tear itself open in its central region, 3, and thus to cause a cavity in 3, or, in short, to prolong the pipe down into 3. Why, then, does the pipe stop just short of this layer?

There are five causes which may co-operate to arrest the down-stretching of the pipe. They are:

(1) The hypothetical possible expansion of the axial metal at the moment of freezing;

(2) The in-pressing of the ingot's sides by the atmospheric pressure;

(3) Blow-holes;

(4) The down-sagging of the metal from above; and

(5) In the lower part of the ingot, the lagging of the solidification of each layer behind that of the layers beneath it, or, in other words, the fact that solidification is proceeding from below upwards.

These five causes we will now take up in §§ 22 to 30.

§ 22. *Cause* (1). *The expansion of the axial metal in the act of solidifying*, might, indeed, contribute to arresting the down-stretching of the pipe at layer VI., and thus to preventing it from entering layer V.

In order to fix our ideas, let us hypothetically assume that steel

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expands near the freezing-point, as sketched in Fig. 10. If this is assumed, then it might well be that the expansion of part 3 of layer V., in the act of freezing and climbing from *c* towards *e* of Fig. 10, contributes more efficiently towards compensating for the simultaneous outward drawing of parts 2 and 4, than the corresponding expansion of the axial part of layer VI. In layer VI. this expansion of the axial part may not have been so well timed for its like work of compensating for the outward drawing of the inner walls.

But this cause in and by itself does not suffice to explain

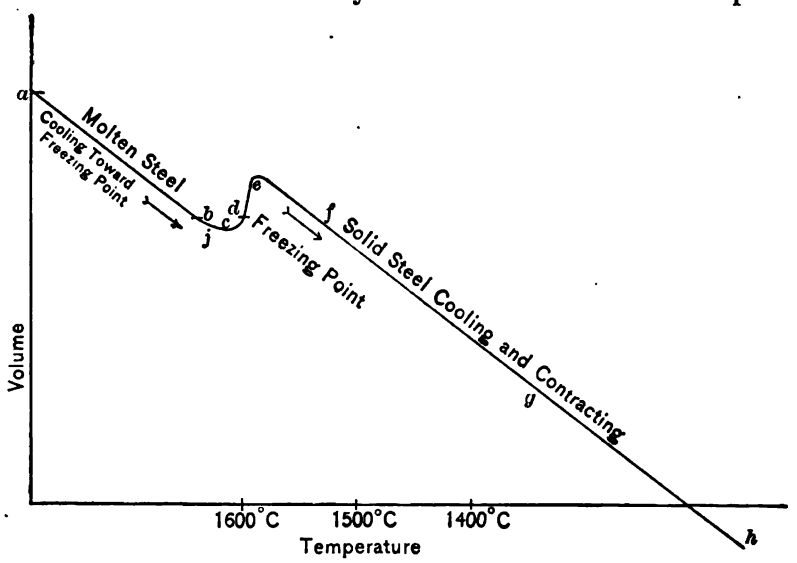

FIG. 10.—VOLUME-TEMPERATURE CURVE OF STEEL, ON THE HYPOTHESIS THAT IT EXPANDS IN PASSING THE FREEZING-POINT.
This hypothesis is not adopted in the present paper.

why the pipe does not stretch into layer V., nor, indeed, why it does not, for like reasons, stretch away down to the bottom of the ingot. There are two reasons why it fails fully to explain the arrest of the pipe. The first of these is that at the time when part 3 has passed the crest *e* of Fig. 10, and has again begun to contract with its further cooling, the metal there is still so weak and mushy that it ought to draw apart under the continued outward stress due to the continuing outward pull of parts 2 and 4; so that after all a cavity ought to form in this layer V., or, in short, that the pipe ought to reach down into this layer. And what is true of this layer ought, by like reason-

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ing, to be true of each following layer downwards towards the bottom of the ingot. So that, wherever the pipe happens to end, the question remains, " Why did it not go one step deeper down? "

A second even more cogent reason why the expansion of the metal in freezing does not suffice to explain fully the arrest of the pipe, is that, whereas nearly all substances, so far as I have noticed, pipe in freezing, yet in none of them does the pipe stretch down nearly to the bottom of the ingot, unless, indeed, the conditions are unusually favorable to its down-stretching. Now, unless we are prepared to go to the length of assuming that all of these varied substances—metals, alloys, slags of all kinds, wax, paraffine, sulphur, and what not—also expand at the moment of solidification, this explanation breaks down, and some further explanation is needed.

In short, the expansion of gray cast-iron and ice at the moment of solidification may help to arrest the down-stretching of the pipe in these substances; but we must look to some additional explanation for this arrest in the miscellaneous substances which, though piping, yet like cast-iron do not pipe to an extreme depth, and unlike cast-iron do not expand in freezing, and this additional explanation should apply to ice and cast-iron also.

§ 23. *Cause* (2). *May the atmospheric pressure help to arrest the down-stretching of the pipe?* In case of an ingot like that shown in Fig. 11, the tendency of the pipe to stretch down below the lowest point which it actually reaches may have been restrained by the bulging-in of the walls of the ingot from atmospheric pressure. This is easily understood in this particular case, because the pipe is surrounded in every direction, above and below as well as on its sides, by a thickness of solid metal too great to permit any rapid infiltering of atmospheric air to fill the vacuum which the gaping of the pipe tends to cause. Of course, this space could not in any case be a true vacuum, because the gas dissolved in the steel would evolve freely into it. But yet this evolution of gas might be so slow that the actual pressure in the pipe would be very far below the atmospheric pressure on the outer walls of the ingot; and the excess of the outer atmospheric pressure over the pressure of gas within the

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pipe might press the walls of the ingot in enough to arrest the down-stretching of the pipe.

In other cases we cannot feel so sure that the atmospheric pressure does contribute materially to this arrest of the pipe. For instance, in the ingots shown at *A* of Fig. 22 and in Fig. 12, the crust at the top of the ingot is so thin that the air might either filter in through holes in it or might press in the top itself enough to raise the pressure in the

Taken from Ledebur, *Eisenhüttenkunde*, vol. iii., p. 864, Fig. 287.

FIG. 11.—STEEP-WALLED PIPE IN A STEEL INGOT.

The steepness of the walls of this pipe indicates that it opened after the steel had reached an advanced degree of firmness. The obliqueness of the blow-holes at the sides suggests that they have formed in plastic metal. The pear shape of those at the bottom suggests that their upper part has broken away and floated up towards the top of the ingot. It may be represented by the irregular blow-holes in the upper part of the ingot.

FIG. 12.—VERY NARROW STEEL INGOTS PIPE DEEPLY.

The general smoothness and bell shape of the upper part of the pipe suggest that it formed when the remaining unfrozen metal was decidedly mobile, though of course pasty at its very shores, where freezing was actually taking place. The slight furrows in this part seem to have been formed by the down-sagging of the lately deposited shore-layers as soon as they were left unsupported by the ebb of the tide. The steepness, crystal-facing, and at last discontinuousness of the lower part of the pipe suggest that it formed after the last of the metal had become relatively firm.

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pipe nearly to the atmospheric pressure. Again, in the zinc ingots in Fig. 15, and in many steel ingots (see § 38), the pipe is open to the air, so that the outward atmospheric pressure on the sides of the pipe should equal the inward atmospheric pressure on the outer walls of the ingot.

Thus the atmospheric pressure, while it may often help to arrest the down-stretching of the pipe, does not suffice by itself to explain why the pipe usually stops so high up in the ingot, even when there are no blow-holes.

§ 24. *Cause* (3). *That blow-holes should lessen the pipe* will be self-evident to many. For those to whom it is not, let me add that these blow-holes are permanent bubbles which form within the plastic inner walls of the freezing ingot. In so forming they must in effect puff out and enlarge or thicken those walls; and we have seen that whatever enlarges the inner walls thereby pushes their shores nearer together and thereby lessens the space between those shores, in which space lies the molten lake. But to decrease this space must raise the upper surface of the lake, or at least must restrict its ebb, and must thus lessen the pipe which is the cavity left vacant by that ebb. (See §§ 48 and 49.)

But this does not help to explain why, in ingots of metal apparently wholly free from blow-holes, the down-stretching of the pipe is arrested very far above the bottom of the ingot. Clearly, we are no nearer the end of our search than we were at the end of § 22.

§ 25. *Cause* (4). *Sagging arrests the down-stretching of the pipe.* And this brings us to the fourth of our explanations of the arrest of the pipe, an explanation which, from its very simplicity, might easily be overlooked, the down-sagging of the lately-frozen parts, at each period during the freezing. This sagging tends to fill up any incipient crevice in any given horizontal layer as fast as it starts to form, to fill it with molten metal as long as the metal just above that crevice remains molten, and afterwards with pasty metal.

To clarify our ideas, let us follow the freezing of a given layer, say layer I. of ingot *E* in Fig. 1, p. 6. At the moment when the axial part of this layer, 8, is freezing, and tends to neck or to gape because of the outward movement of parts 7 and 9, the axial matter just above is still molten and runs down to meet

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this necking, gaping or even cracking, moved down not only by its own weight but also by the downward pressure of all the still molten and pasty metal above. And, when even the axial matter in layer II. has frozen, it is still so soft and pasty that the pressure of the molten and pasty metal above presses it down to meet the necking and gaping tendency in the axis of layer I. Thus what really happens is that the axial part of each layer is (1) tending to tear or gape open because of the outward drawing of the parts to right and left of it, and is at the same time (2) forced down to replace the metal which like causes are removing from the axial part of the layer next below, and (3) is receiving fresh metal from the axial part of the layer above, which in turn is pressed down by the weight of all the molten and plastic metal above it. Thus the metal, which was initially in a given horizontal layer, is by this action made to sag somewhat, as is sketched in layer III. The essence of this explanation, then, is that the tendency of each layer to axial gaping, or the gaping tendency, is aggravated by its need of feeding metal down to meet the gaping in the layer beneath, and met by its receiving metal from above.

Whether the pipe shall stretch down into a given layer, then, depends upon whether the overlying molten or pasty metal is fed down fast enough to satisfy the gaping tendency in the axis of that layer. This gaping tendency may be called the pipe-forming tendency, and the down-sagging tendency of the molten or pasty metal the pipe-closing tendency, a tendency to which, as we saw in § 24, the formation of blow-holes contributes. Clearly, as we pass up from layer to layer, the supply of molten and pasty metal from above becomes smaller and smaller, and the weight of the metal above which forces it down becomes less and less. The very lowest layer into which the pipe, in the cold ingot, is found to reach (layer VI. of Fig. 1, *E*) is that in which the supply of molten and pasty steel from above has not sufficed to overcome the gaping tendency; in other words, to make good the withdrawal of the axial metal outwards and downwards, up to the time when that axial metal has become so cool, solid and firm as to endure the outward drawing stress without breaking. The next layer below, V., has received sufficient strength, before the supply of metal from above failed, to endure the outward stress without breaking.

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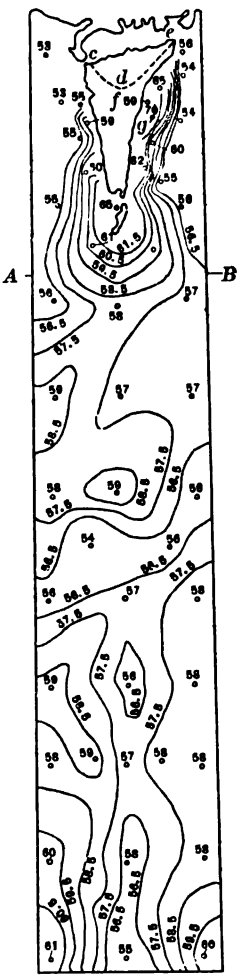
If we assume that the axis of layer IV. of ingot *E*, Fig. 1, is the hottest spot in the ingot, so that it is the last to freeze, the isotherms will be grouped onion-wise around it somewhat as sketched. When the axis of this layer freezes, there will be no molten metal above it to run down freely to meet its gaping; but instead pasty metal may be fed into it under the pressure from above. If, because the ingot is thick, or because its mold has been preheated, or for any other reason the piping tendency and the outward-drawing of layer IV. are small, the down-sagging of the pasty metal from above may meet the gaping tendency in this layer until its axial metal has grown strong enough not to break under that outward-drawing stress; or, in short, the pipe may not reach to this the last-freezing point, the richest of the segregation. If, on the other hand, the piping tendency is strong, and the supply of pasty metal from above is small, as in thin ingots like those of Fig. 22, not only this layer but even those far below it may fissure, because in them the supply of pasty metal from above falls short of the demand made by the gaping tendency, at a time when the axial metal is still too weak to endure the outward-drawing stress, so that the pipe may stretch far deeper than the richest point of the segregate.

In most of the cases which I have examined, the pipe does not reach down as far as the richest of the segregate—*i.e.*, as far as the last-freezing spot; but in a case reported by Stevenson and Kent,[6] reproduced in Fig. 13, it apparently reaches far



(A. A. Stevenson and R. Kent, *Trans.*, xxiii., 637.)

FIG. 13.—PIPING, SEGREGATION, AND ISOCARBS IN A STEEL INGOT.

The contour-lines here drawn give the approximate position of the carbon-contents which they represent. The numerals in this figure are the carbon-contents, in hundredths of 1 per cent.

[6] *Trans.*, xxiii., 637 (1893).

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below this spot. To show the progress of the solidification and the progressive enrichment by segregation, I have plotted in this the lines of equal carbon, or "isocarbs."

§ 26. *Evidence of this sagging action*, which I have been forced to introduce in order to explain why the pipe does not stretch nearly to the bottom of the ingot, is supplied by the nearly vertical furrows, which often line the upper part of the pipe, as shown in Fig. 14, and less distinctly in the upper part of Fig. 12. Each of these furrows is in a nearly vertical plane, which passes nearly through the axis of the ingot, and, taking them

FIG. 14.—THE FURROWS IN THE UPPER PART OF THE PIPE IN AN 8-IN. STEEL INGOT IN THE AUTHOR'S COLLECTION.

as a whole, they are such furrows or tracks as should result from the downward sliding of the viscous, partly frozen steel which the ebb leaves exposed on the shores. As this viscous steel slides down, the progressive narrowing of the converging walls over which it slides should tend to pucker it, thus accenting this furrowing.

One may not speak with perfect confidence about the cause of these furrows. In many cases their shape could be explained by simple puckering due to horizontal contraction; but in the ingot from which Fig. 14 is taken the indications of sagging seemed to me irresistible, after careful study.

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§ 27. *Need of an additional explanation.* The four causes which we have now considered: (1) possible expansion of the axial metal in solidifying; (2) the atmospheric pressure; (3) blow-holes, and (4) the down-sagging of the axial metal, do not seem to explain easily why the pipe is so short in an ingot of zinc, such as is shown in Fig. 15, I. and II. Here and in like cases I see

I. II. III.

FIG. 15.—INGOTS OF METALLIC ZINC, PREPARED BY DR. WILLIAM CAMPBELL IN THE AUTHOR'S LABORATORY.

The three ingots here shown have been sawn vertically through their axes, so as to show the pipe. Ingot I. has been etched, in order to show the columnar structure. The obliqueness of this structure in the lower part of the ingot shows the cooling effect of the bottom of the mold. Ingots II. and III. were made in the same way, except that the large end was lowermost in II. and uppermost in III. This difference has lengthened the pipe greatly in ingot II. by making the freezing take place from above downwards, and thus impeding the down-sagging of the axial metal to fill the nascent pipe.

no reason to suppose (1) that the metal expands in solidifying, especially because the pipe is wholly open at the top. This very openness, moreover, seems to exclude the idea (2) that the atmospheric pressure on the outer sides of the ingot may have arrested the down-stretching of the pipe; (3) no blow-holes are to be seen, and invisible, imaginary blow-holes are not a wholly satisfying explanation of the fact that the pipe does not stretch

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down as deep as we should expect. Finally, (4) zinc passes so suddenly from the molten to the solid state that we are loath to believe that down-sagging can have aided materially to arrest the down-growth of the pipe. Indeed, the columnar crystals (Fig. 15, I.) developed by etching give no strong evidence of any such sagging.

It was with the purpose of learning whether these four causes suffice to explain the arrest of the pipe that I requested Dr. William Campbell to prepare for me ingots of a substance which passes abruptly from liquidity to solidity, so that we could see whether in these the pipe was not thus arrested but stretched nearly to the bottom. The fact that it is thus arrested, even in this case in which it ought to stretch deep, showed me that an additional explanation was needed, and thus called my attention to the fifth of our causes, which we will now consider. I have to thank Dr. Campbell for his care in making and etching these ingots for me.

§ 28. *Cause* (5). *In the lower part of the ingot, the fact that solidification proceeds from below upwards restrains the down-stretching of the pipe.* In an ingot like III. of Fig. 15, or *B* of Fig. 22, the solidification of the lower part, as a whole, should outrun that of the upper part for three distinct reasons: first, because the lower part is poured into the mold before the upper part, so that its cooling and solidification should begin earlier than those of the upper part; second, because the lower part is narrower than the upper; and third, because the cold "stool" or bottom of the mold greatly hastens the cooling of the lower part of the ingot. As the solidification of each horizontal layer thus lags behind that of the layers beneath it, so is each layer the better able to feed down and supply the necking or gaping in the layer beneath.

It is wholly in accord with this idea that when, as in ingots *A* of Fig. 22, and I. and II. of Fig. 15, cast with the large end down, the second of these reasons is lacking, the pipe stretches far deeper than in those cast with the large end up, a matter which we will consider further in § 42. All this goes to show that the more the freezing of the upper layers lags behind that of those beneath, the more efficiently do the upper layers feed down and fill up the gaping in those beneath, and the shorter is the pipe.

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Though this is true of the ingot taken as a whole, in the upper part of the ingot this lag is lessened by the fact that here much heat escapes upwards through the upper surface in addition to that which escapes laterally into the mold-walls. Solidification instead of taking place from below upwards here takes place from above downwards. The very uppermost layers, instead of lagging behind those beneath them, outrun them; instead of feeding down to fill the gaping in the layers beneath, these upper layers very often actually freeze across so as to form a bridge too rigid to bend down to meet the gaping and sagging beneath the crust at the top of the ingot.

At the lower end of the ingot the opposite state of affairs exists. Here the rapid escape of heat into the stool which forms the bottom of the mold greatly hastens the cooling of the very bottom, making it outrun greatly that of the metal above, or, looking at it in the opposite way, greatly increases the amount by which the cooling and freezing of each layer lag behind those of the layer beneath, and thereby favors the downward feeding of each layer to fill up the gap in the layer beneath. Moreover, this lag becomes more and more pronounced as we get nearer and nearer to the bottom of the ingot, and the cooling effect of the bottom of the mold becomes felt more and more. Hence, the nearer we get to the very bottom of the ingot, the more fluid will each layer be during the outward-drawing and gaping of the layer beneath, and the better, therefore, should each layer fill the gaping of that beneath, and, in fine, the more powerfully is the formation and down-stretching of the pipe opposed.

Thus it comes about that under no conditions does the pipe ever reach to the very bottom of the ingot. Even under the extremely favorable conditions of bottom-casting, to be described in § 40, the pipe is at least always closed at its lower end.

The effect of this downward flow of heat into the bottom of the mold is clearly seen in Fig. 15, I. Here, throughout the lower 25 or 30 per cent. of the ingot's length, the inner ends of the columnar crystals developed by etching point upwards, showing that the downward flow of heat had been strong enough to influence their orientation greatly. Of course, a downward flow of heat implies that the cooling of each layer lags behind

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that of the layers beneath, because heat will flow only from a hotter to a cooler object.

The cooling-action of the stool is much stronger than that of the walls of the mold proper, first, because the stool is usually very much thicker than the mold-walls, and second, because it is always and necessarily in firm contact with the ingot, which is pressed down upon it by gravity, whereas between the mold-walls and the ingot an empty air-space opens early in the freezing, due to the horizontal contraction of the cooling ingot and the simultaneous horizontal expansion of the rapidly-heating mold.

§ 29. *Evidence of the importance of cause* (5) is given by the results of bottom-casting, which, I believe, is the only condition under which this lag of the cooling of each horizontal layer behind that of the underlying layers is artificially prevented in the greater part of the lower end of the ingot. How it is prevented we shall see in § 40. In other words, suffice it here to point out that bottom-casting leads to concentrating the heat at the bottom of the ingot instead of at the top, so that it can be only in a very short region close to the bottom that the solidification proceeds from below upwards. Throughout nearly the whole length of the ingot it comes about that solidification progresses in the opposite fashion, from above downwards, so that each layer, instead of being in a position to feed down and fill the gaping in the layer beneath, freezes earlier than that beneath, and is thus prevented from filling up the gaping there. In short, this solidification from above downwards ought to promote the down-stretching of the pipe, by preventing cause (5) from taking effect, and this it certainly does in bottom-casting.

Thus it came about that those who practiced bottom-casting, whether they understood the reason or not, actually found that, when carried out effectively, it tended to cause an extremely deep pipe. An illustrious Pittsburg steel-maker is said to have summed this up with the remark that, after he had perfected bottom-casting so fully that his ingots piped right through from top to bottom, he abandoned it. Fig. 16 shows, on Ledebur's authority,[7] a bottom-cast ingot which has thus piped nearly to the bottom. The stumpiness of plate-ingots, together with the presence of blow-holes, may counteract this piping.

[7] *Eisenhüttenkunde*, 3d ed., vol. iii., p. 862 (1900).

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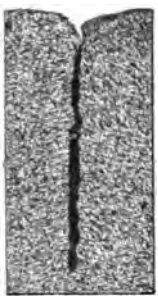
§ 30. *Summary of the five causes which arrest the down-stretching of the pipe.* Expansion in freezing, atmospheric pressure, and blow-holes may contribute, with varying degrees of effect in special cases, to aid the arrest of the down-stretching of the pipe; but neither severally nor jointly do they suffice to explain its arrest in cases like that shown in Fig. 15, I and II. But the down-sagging of the molten and pasty metal, and in the lower part of the ingot the progress of solidification from below upwards, or, in other words, the lag of the cooling of each layer behind that of the underlying layers, are causes which not only are at work in nearly all cases, but seem competent in all cases to explain this arrest, especially in view of the fact that when, as in bottom-casting, this progress is from above downwards throughout nearly the whole length of the ingot, then, and so far as I know, then only, may the pipe stretch nearly to the bottom.



Taken from Ledebur, *Eisenhüttenkunde*, vol. iii., p. 862, 1900.

FIG. 16.—VERY DEEP PIPING, SUCH AS MAY RESULT FROM BOTTOM-CASTING OR VERY RAPID CASTING.

In considering these cases, we should remember that the pipe probably stretches down below the point at which our section finds it, because the lower end of the pipe is very narrow, and may not be quite continuous, and a vertical section through an ingot, first, may not happen to pass through the very lowest part of the pipe, and, second, if it does, it is likely to obliterate this lower part. For in cutting the ingot open we have to use not a mathematical plane but a cutting-tool of some appreciable thickness; and this tool may easily obliterate all traces of those parts of the pipe which are narrower than the thickness of the tool itself.

§ 31. *How ought sagging to affect the shape of the pipe?* In our first sketch of the formation of the pipe, Fig. 1, for simplicity of presentation we ignored sagging, and, indeed, the important fact that the metal remains soft, pasty and flowing for a considerable range below the point at which freezing begins. Let us now see how the ideas which we there formed are to be modified by these facts.

Let us assume that at an early stage in the ebb of the molten

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lake, the pipe has the shape *ABCD*, shown in Fig. 17. Here *BE* represents the left-hand shore or wall of the submerged part of the molten lake, or the boundary between the molten and the solid part. When the lake has ebbed to the position *FG*, shown in broken lines, the support which in the stage *ABCD* was offered to the part *BE* of the walls by the molten steel, has now been withdrawn, and therefore the metal thus left unsupported tends to flow down towards the cavity thus left, so that its upper surface sinks from its former position, *AB*, to something like *AF*. When the lake has ebbed further, to *JK*, the support which, in the second stage, the walls *FH* received from the molten metal at their right, has now in turn been withdrawn, and further sagging takes place, in which both the part *FH* and the previously bared part, *AF*, should

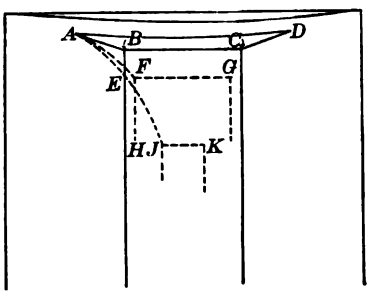

FIG. 17.—THE EBB OF THE TIDE LEAVES THE SHORES OF THE PIPE UNSUPPORTED, AND THUS LEADS TO SAGGING.

share, so that the lower side of the pipe should now have a shape something like *AEJ*.

§ 32. *How far is the pipe due to sagging rather than ebbing? Discussion.* Carrying this idea still further, step by step, we come to ask ourselves whether this sagging after solidification may not account for the whole of the pipe, and whether our first idea that it forms with the ebb of the molten lake may not be wrong. In short, may it not be that the pipe forms after even the axial layers have solidified, not by the ebb of a molten lake but by the sagging of pasty metal distinctly below its melting-point? Thus, to let the extreme case illustrate the principle, let us assume that no crevice forms within the ingot until even the axial metal has begun to freeze, and is in a mushy or pasty stage between true liquidity and effective rigidity, when it neither runs like a typical liquid nor stands firm

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like a typical solid; and that at this time a vertical crevice, *KLM*, Fig. 18, opens in the axis of the ingot, thanks to the outward drawing of the interior metal towards the outer walls. That such an initial shape is possible, the slit-like pipe in Fig. 11 shows. Would not the sagging of the pasty metal surrounding this crevice, combined with the outward drawing of its walls, change its shape through the successive stages, *NP*, *N'P'* and *N''P''* (of the first two I sketch only the upper part), to the typical pipe-shape, *QRS*?

It might, if we make one reasonable admission in order to meet a difficulty which at once confronts us. This difficulty is, that if the metal in the relatively cool part between *N''* and *S* were hot and soft enough to sag at all, in short, were so weak

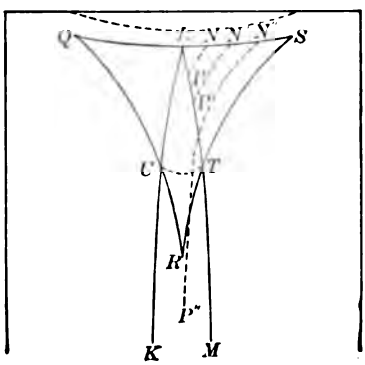

FIG. 18.—CAN THE BELL-SHAPE OF THE PIPE BE DUE WHOLLY TO SAGGING?

that gravity sufficed to overcome its cohesion with the roof of the pipe which we suppose it to leave, then the metal in the center and bottom of the pipe, lying as it does in the axis of the ingot, should be so fluid that it should gather at least into a puddle with a meniscus-shaped top, if not into a lakelet with a level top. In short, the bottom of the pipe should be level, or at least rounded, instead of being pointed, as it really is. But this difficulty is readily met. This metal which sags down to the bottom of the crevice might at first readily have an upper surface like *UT*, for of course its most liquid part, running fastest and farthest, would be likely to gather beneath, and the most sluggish of the metal, coming last, would be on top of the more liquid. Now, in the further cooling and outward

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drawing of the walls, this puddle, *URT*, progressively freezing harder and harder, might readily yawn down as a crevice with nearly vertical walls.

Indeed, thus, it seems to me, we must explain the fact that the lower end of the pipe so often has nearly vertical walls. This lower vertical part, so far as I can see, must form, or at least must reach its final shape, after the metal has become distinctly solid, without any great degree of mobility; for otherwise the metal of these vertical walls would necessarily sag down to fill the lower part. For instance, all but the very upper end of the pipe in ingot *A* of Fig. 22 and in Fig. 12, must have formed after even the axial metal had reached an advanced degree of firmness. It is in accord with this idea that, so far as I have noticed, the walls of the lower vertical part are not smoothed or rounded, as if left by the ebb of a molten or even pasty lake, but ragged or crystalline, as if torn open through the solid metal, firm enough to stand thus cliff-like without sagging, yet not so rigid but that its particles can migrate into crystalline forms. This some metals can do at temperatures very far below their melting-points. Silver and copper are credibly reported to migrate into moss-form even at the atmospheric temperature.[8]

Further very strong evidence that the lower part of the pipe sometimes forms in metal which has already grown distinctly solid, is given by its sometimes stretching down far below the richest of the segregate, which is the last point to freeze. For if this, the very last freezing part, had, at the time of the opening of the lower part of the pipe, been liquid or even plastic, it would have flowed down into the chasm which thus opened beneath it; and the fact that the chasm thus yawns down below the last frozen part shows that it must have opened after the last of the freezing had ended, or, in other words, in metal which was already distinctly firm and solid.

In Fig. 13, p. 39, the width of the pipe at the level of the richest of the segregate, *g*, and the great distance to which it stretches below this level, argue powerfully that there must have been a great deal of gaping open and of downward-stretching of the pipe after even the last of the metal had acquired much firmness,

[8] Hutchings, Readwin, and Collins, *Chemical News*, vol. xxxv., pp. 117, 144, 154 (1877).

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because otherwise this richest of the segregate would have run down to the bottom of the pipe, instead of staying perched up high on the side of the chasm which yawned beneath it. In picturing this matter in our minds, we must suppose that the richest of the segregate was originally much above its present site, for instance at *f*, when the lower part of the pipe had a shape somewhat like *cde*. Later the pipe continued to stretch down, but, for some reason hard to guess, apparently by a path which did not pass through the richest of the segregate. As it thus stretched, a certain degree of sagging took place, widening the upper part of the pipe, and carrying certain rich parts of the segregate far from their initial position. Note the very rich spot, 0.65 carbon, near the bottom of the pipe, far below spots close to the sides of the pipe with much less carbon—*e.g.*, 0.59 and 0.60. That the parts which thus slid or sagged had only slight mobility is indicated by the imperfect way in which they closed the bottom of the pipe, leaving irregular unclosed patches at its lower end, as pellets of gradually stiffening tar might, in sliding down such a hole. This conception helps to explain the irregular distribution of the carbon in the neighborhood of the pipe.

The position, *g*, of the richest of the segregate, 0.70 carbon, must not be taken too confidently. Other spots lower down at which drillings were not actually taken, might, if examined, have contained even more than 0.70 carbon. It is even possible that the richest of the segregate may lie below the bottom of the pipe; but in any event the irregularity with which the spots richest in carbon are distributed argues strongly that much yawning open and some sagging must have taken place after the segregation had very nearly completed itself, and when there was no large amount of metal that was still mobile, or when all but a very little was in an advanced stage of solidification.

Returning to the question with which this section opened, while it may be possible that the pipe should begin to form, not before, but after even the axial region has begun freezing, such evidence as is before us indicates that much of it usually forms while the metal is extremely fluid, and as sketched in Fig. 1. Some of the evidence we will now consider.

§ 33. *Evidence furnished by the bridges.* Valuable light on this

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question is given by the bridges which sometimes stretch across the upper part of the pipe in steel ingots, and even more characteristically in those of certain slags. Figs. 19 *A* and 19 *B* show a fragment in my collection from the upper part of the pipe in a potful of a ferrous silicate slag from smelting roasted cupreous pyrites. Here we note a series of three distinct bridges, each nearly level, and each smooth on top, but covered with a beautiful growth of crystals on its lower side. This

FIG. 19 *A*.—CRYSTALS ON THE LOWER SIDE OF A BRIDGE IN AN INGOT OF FERROUS SILICATE SLAG.

is just the condition which we find in the bridges in steel ingots; their upper surface is smooth, but their lower surface is covered with fine pine-tree crystals. Professor Stoughton and I found the same condition in our wax ingots.

The smooth, continuous, level upper surface of such a bridge is good evidence that this surface, when it formed, was for the moment the surface of the molten lake; and the horizontal width of this bridge represents the width of the upper surface of the lake when it was at this same level. That this bridge

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has formed indicates that, for some reason which we need not now stop to consider, perhaps because of some phenomenon of surfusion, the ebb of the lake has been arrested long enough to allow its upper surface, which is radiating heat upwards across the empty space above it to the very top crust of the ingot, and therefore cooling comparatively fast, to freeze across. Such an arrest could easily be caused by the formation of a ring of blow-holes, beginning when the level of the lake had fallen to where we now find the bridge (see § 24). Each bridge seems to form quite as the top crust of the ingot forms, by the freezing of the level surface of a molten lake.

FIG. 19 *B*.—THREE LEVEL BRIDGES FROM AN INGOT OF FERROUS SILICATE SLAG.

The relatively rigid slag bridges, such as are shown in Fig. 19 *B*, preserve their original evenness; they remain smooth and level; yet the bridges in a steel ingot often sag. But then this sagging is only natural, and is like the sagging of the top crust of the ingot. We know with absolute certainty that this latter forms by the freezing across of the upper surface of a molten lake; its sagging is only the natural behavior of this metal, which remains soft and plastic for a considerable range below its freezing-point, so that, as the lake beneath it ebbs, gravity bends down the originally level bridge which this ebb leaves unsupported.

It is quite in conformity with this picture that often the bridge

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in steel ingots is torn away from one of the sides of the pipe, and, thus left unsupported at that end, sags down there by its own weight. This is the case with the right-hand end of the bridge in the ingot from which *E* of Fig. 1 has been sketched, and it means that in their outward drawing the inner walls of the ingot have stretched the bridge until it broke away from one of its bearings. That it should thus break is readily understood, because, spanning as it does the chasm above the surface of the lake, its cooling outruns that of the remainder of the ingot in its neighborhood; for through the empty space

FIG. 20.—A BRIDGE IN THE PIPE IN A STEEL INGOT, BROKEN AWAY FROM ITS SUPPORTS.

above it the bridge radiates heat rapidly to the thin cool upper crust of the ingot. Cooling faster, and therefore contracting faster, than the neighboring metal, it breaks away from one of its bearings, as a guitar string snaps when stretched too tight.

In an ingot which has been broken open forcibly by severe blows, I have sometimes found that the bridge has broken completely away from the sides of the pipe, and lies within it as a detached disk. In Fig. 20 the bridge is seen to be almost completely detached. Of course, the severe shock which such an ingot undergoes when it is broken open may well increase the degree to which the bridge is detached.

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The beginning of the formation of such a bridge is shown at level *A* in Fig. 21; and between the main cavity and the top crust are seen two distinct bridges. In our wax ingots Professor Stoughton and I found a very great number of these bridges, one above another.

§ 34. *Evidence given by the lower side of the bridges.* The condition of the under side of the bridge, not less than the existence of the bridge, tends to show that it formed the upper crust which had frozen across the top of the molten lake when it had

FIG. 21.—STALACTITES ON THE LOWER SIDE OF A BRIDGE IN A STEEL INGOT.

ebbed to this level. The stalactites hanging from beneath the main bridge in Fig. 21 have evidently been left by the sinking down of the molten lake which once wetted the under side of this bridge, and therefore was spanned over by it.

The crystals which more often crust the lower side of the bridge evidently have formed in the molten metal in contact with its under side, and have been left exposed by the ebb of that metal, or, as in case of Fig. 19, of that slag. We naturally ask how it comes that these beautifully fine crystalline

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faces, sometimes with pine-tree growths as fine as hairs, are thus clear and sharp cut instead of being rounded like stalactites, for example, like those in Fig. 21. If, for instance, we were to dip the crystal-faced bridge of Fig. 19 *A* into a bath of slag and remove it, and allow that molten slag to drip off, or if we were to dip it in water, withdraw it, and immediately expose it to a temperature below 0°C., those exquisite crystalline facings would be smoothed over and obliterated by the frozen remains of the once-liquid bath into which we thus dipped them; for this liquid could not drip off so fully as to leave the crystalline faces clean.

The answer is, that as the ebb leaves a given very small layer bare and wet with the drip of the ebb itself, that drip is forced by the nascent crystal on which it is left into crystalline unity with that crystal. The molecules of the drip are polarized by the crystalline force into adopting the shape of the crystal itself and merging themselves in that crystalline growth, quite as when a crystal grows within a quiescent mother-liquor, this same crystalline force dragoons the molecules of the liquid in contact with the growing crystal to identify themselves with that crystal and form part of its growth.[9]

But the fineness and sharpness of the pine-tree crystals which incrust the lower side of the bridge, and the distance to

[9] I must confess that, although this explanation is on the whole satisfying, yet it leaves a slight difficulty yet unexplained. When the crystal grows in the mother-liquor it grows by selection. Quite as the plant selects from the earth that which it needs and rejects that which it needs not, so does the growing crystal select from the mother-liquor the matter which is like to itself and rejects into that mother-liquor the unlike matter. In short, this growth is by selection and rejection. But when the ebb leaves on the lowest projecting point of a downward-reaching promontory-like crystal a last drop of molten mother-slag, this drop should be parted by the crystal on and into which it grows into two parts: matter like the crystal itself, which is selected and merges with the crystal, and matter which is unlike and is rejected. But what has become of this rejected matter? We do not find it on the lower edge of the crystal, which is as sharp and sometimes as thin as a sheet of note paper. It cannot have fallen off without leaving a residual droplet; but none such is found. Does this not mean that the crystal has the power to force into its own shape and growth the last particle of mother-slag or mother-metal, even though this differs slightly in composition from the crystal itself? Does it not also mean that those relatively rare cases in which sharp, clear-cut crystals line a cavity or vug, are just those in which the mother-liquor does differ so slightly in crystalline behavior from the crystals which form within it, that those crystals are able to force the drip thus left to form part of their growth?

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which they sometimes shoot out into the space beneath, show that that tide, the ebb of which has left them bare, must have been very liquid. The down-sagging of a pasty magma could hardly have left these forms so clear, so sharp, and especially so far outstretching into the void. This applies with great force to the large ill-shaped crystals which are sometimes found on the walls of the pipe beneath the bridge. It is hard to resist the conviction that the ebb which left these bare must have been of a decidedly mobile liquid, not of a barely viscous solid.

§ 35. *Evidence given by the surface of the pipe.* The general smoothness of the surface of the upper part of the pipe, even when it is furrowed as shown in Fig. 14, is that of a surface left bare by the ebb of a molten or at least of a very mobile lake. Such smoothness could hardly result from tearing open a crevice in a solid which has any considerable firmness; but it is what we should naturally expect from the ebb of a molten lake, especially when we remember that the very edges of the lake are at the freezing-point, and that while it is ebbing its layers next its shores are freezing.

§ 36. *Summary.* To sum this up: if the pipe is prolonged downwards with very steep, cliff-like, nearly parallel walls, this part seems, by its steepness and the sharp-pointing of its lower end, to have been opened after even the axial metal here has become so distinctly solid that it could not flow down to fill this crevice. But the relative flatness, the bell-shape of the upper part of the pipe itself, the smoothness and levelness of the upper side of the bridges, the sharpness of the crystals which sometimes incrust their lower side, the far out-shooting of crystals beneath them into the pipe itself, and the smoothness of the upper part of the pipe-walls, as contrasted with the crystalline mossiness of the steep lower walls, seem to show irresistibly that this upper part is opened early during the freezing, when the metal which it replaces is distinctly molten. After this upper part is thus opened, it may be further widened by the outward drawing of its shores towards the outer crust of the ingot to which they are welded.

§ 37. *How to shorten the pipe.* It is very important to shorten the pipe as much as we can, so that that part of the ingot which has to be rejected because it contains the pipe may be as short as practicable. As we saw in § 21, there is the strug-

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gle between the pipe-forming or gaping tendency in the axis of each layer, due to the outward drawing of the metal towards the outer walls, which in turn is due to the virtual expansion of those walls, and the pipe-closing or sagging tendency, reinforced by the growth of blow-holes. The depth to which the pipe reaches depends upon the relation between these two tendencies, the outward-drawing, gaping or pipe-forming tendency, and the down-sagging or pipe-closing tendency. We may shorten the pipe by lessening the pipe-forming tendency, or by increasing the pipe-closing tendency, or both. Let us consider these two methods separately.

§ 38. *Shortening the pipe by lessening the virtual expansion of the crust.* If the cooling of all parts could proceed at exactly the same rate, so that the several concentric layers contracted at the same rate, clearly no pipe would arise, and the outer layers, like all the others, would, at the end of the cooling, reach exactly their natural dimensions. It is because of the early virtual expansion of the outer walls that the inner layers later draw outwards towards those outer walls, that the gaping or pipe-forming tendency arises in each successive layer, and that, at the end of the freezing, the interior does not suffice to fill the virtually expanded outer walls, and, in short, that the pipe forms. The volume of the pipe should represent the amount of this virtual expansion, and the greater this virtual expansion the greater should be the pipe. If, for instance, at the moment when the outer walls have cooled to temperature *f* in Fig. 10, p. 34, the metal within them is still so hot and expanded as to occupy volume *e*, then the outer walls are virtually expanded by an amount corresponding to the vertical distance between *f* and *e*. In short, the more the contraction of the interior lags behind that of the outer walls, or, to put it conversely, the more the cooling of the outer walls outruns that of the interior, the greater will be the virtual expansion of those outer walls, and the greater consequently will be the pipe.

Now what causes the outer walls to outrun the interior in cooling is the rapid removal of their heat by and through the mold-walls. If the mold-walls are thick, and the ingot relatively narrow, they abstract heat from the outer shell of the ingot much faster than that heat can be replaced from the interior of the ingot; so that the degree of outrunning is very

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great, the outward drawing tendency and its consequence, the gaping or pipe-forming tendency, are great, and the pipe occupies a large proportion of the volume of the ingot. If, keeping the mold-walls of the same thickness, we make the ingot much wider, then the heat abstracted by the mold is quickly replaced by that which works outwards from the interior of the ingot; the mold becomes heated rapidly, and therefore abstracts further heat from the ingot but slowly. The outer shell of the ingot consequently cools relatively slowly, and its cooling outruns that of the interior relatively little, so that there is but little virtual expansion, followed by little outward drawing, little gaping tendency, and consequently but little pipe.

This agrees with the observed facts that the pipe runs very deep in narrow ingots, such as that shown in Fig. 12, p. 36; that it is large in materials which, like manganese steel, conduct heat very slowly, so that the heat abstracted from the shell of the ingot by the mold-walls is replaced only very slowly from within; that, consequently, it is very small in metals which, like copper, conduct heat rapidly; and that the pipe is shortened by the use of sand-lined molds, and still more by that of preheated sand-lined molds. It agrees also with the fact that, in narrow steel ingots, such as are made from one or two crucibleфuls in the crucible process (the common one-pot and two-pot ingots), there is rarely a solid crust above the pipe, which is open at the top, as in the zinc ingots of Fig. 15;[10] whereas in larger ingots, no matter by what process the steel is made, there is usually a solid crust or bridge across the top of the pipe. In case of narrow ingots the relatively thick mold-walls remove the heat so very quickly, and hence the pipeless period of virtual expansion is passed through so quickly and is succeeded so soon by the piping period, that this begins and makes the molten interior sink down before the cooling-effect of the air has had time to freeze the upper surface across; whereas in large ingots the sinking of the upper surface of the molten steel is delayed long enough to permit this surface to freeze across. (See §§ 5, 14.)

So, too, experiments in my laboratory showed that ingots of copper containing 4 per cent. of silver when cooled suddenly

[10] William Metcalf, Esq., private communication, May 12, 1906.

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piped very deeply, but when cooled slowly formed no important pipe;[11] and that ingots of wax piped much more deeply when cooled suddenly than when cooled slowly.[12]

To sum this up, the pipe may be lessened and shortened:

(1) By casting in wide instead of narrow ingots, and

(2) By casting in sand- or clay-lined molds, especially if these are pre-heated.

§ 39. *Shortening the pipe by increasing the down-sagging pipe-closing tendency.* The preceding sections, and especially § 21, should make it clear that the down-sagging pipe-closing tendency is strengthened by retarding the cooling and solidification of the upper part of the ingot, making these lag behind those of the lower part as much as practicable, in order that, as each layer in cooling from the freezing-point down tends to gape, the overlying metal may be as soft and flowing as practicable, so that it may be forced readily by gravity to sag down and meet this gaping tendency. The more the cooling and stiffening of the upper part lag behind those of the lower, the higher up will lie that layer, the gaping tendency of which is not fully met by the downflow and sagging of the next higher layer; *i.e.*, the higher up will lie the layer in which lies the bottom of the pipe: or, in brief, the shorter will be the pipe.

As we shall see in § 51, p. 75, another beneficial effect of increasing the lagging of the cooling of the upper part of the ingot is to raise the segregation towards the top of the ingot, and thus to lessen the quantity of metal which must be cropped off in order to remove the harmfully segregated part.

This lagging of the cooling of the upper part may be increased:

(3) By top-casting;

(4) By slow casting;

(5) By casting with the large end up; and

(6) By keeping the top hot by means of a sinking-head or an equivalent device.

(A) Segregation further aids in making the cooling of the top lag, and (B) the lagging and downward-feeding are much more efficient in wide than in narrow ingots. We will now consider these six matters in series, and in §§ 48 and 49 we

[11] Experiments by Dr. E. F. Kern and R. C. Blanchard under my direction.

[12] Experiments by Prof. B. Stoughton and myself, to be described soon.

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will briefly consider lessening the pipe by favoring the formation of blow-holes.

§ 40. *Top-casting* is pouring the steel in through the top of the ingot-mold, as distinguished from bottom-casting, or introducing it through a relatively small hole in the bottom of the mold. Whether the pouring is from top or bottom, at the moment when the pouring ceases the bottom of the mold must be hotter than the top, because the bottom of the mold will necessarily have been in contact with molten steel from the beginning to the end of the pouring, whereas the top of the mold does not have that contact until the end of the pouring. So far, then, top-casting and bottom-casting stand on the same footing. But, beyond this, top-pouring favors the concentration of the heat at the top of the ingot, bottom-pouring favors its concentration at the bottom. While pouring the lower part, say the lower quarter of the ingot, the difference in conditions between top- and bottom-pouring is not so very great, because in either case the metal now at the bottom of the mold is pretty thoroughly mixed up by the incoming stream, whether this is falling from the top of the mold or gushing up through its bottom. But henceforth there is a difference, which grows greater and greater as the pouring continues; and this difference is (1) that in bottom-pouring it is the bottom of the mold and ingot that is heated directly by the fresh lots of steel hot from the casting-ladle, whereas in top-casting it is the upper part that is thus chiefly heated; and (2) that in bottom-casting the steel which reaches the top of the mold has been materially cooled off in its upward journey, in which it has transferred much of its heat to the lower part of the mold-walls, and the steel at the bottom is that last added hot from the ladle, whereas in top-casting there has been no such transfer of heat from the steel now at the top of the ingot to the lower part of the mold, and the hottest steel fresh from the ladle is at the top.

This difference increases in importance as the pouring of the ingot progresses, and the steel enters more and more slowly, and consequently stays nearer and nearer to the place where it enters. Be it remembered that the steel keeps much hotter in the great mass in which it lies in the hot clay-lined ladle, under its blanket of slag, than in the molds.

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The sum of this is that, at the moment when the last of the steel has been poured into the mold, in bottom-casting the fresh hot steel is at the bottom and the bottom of the mold has been specially heated, while the steel at the top has been specially cooled; in top-casting there has been no such special heating of the bottom of the mold and cooling of the top of the steel, and the fresh hot steel is at the top.

§ 41. *Slow pouring* favors the lagging of the cooling of the upper part of a top-poured ingot, by prolonging the time during which the steel for the ingot-top is kept hot in the ladle, while the steel already in the mold, and therefore to form the bottom of the ingot, is cooling off and, indeed, solidifying because of its contact with the initially cold mold and stool. Moreover, quick pouring by means of a swift and large stream of molten steel keeps the whole of the metal in the mold in such rapid motion as to efface, in large part, the cooling-off of the lower part of the ingot during the time when the upper part is pouring; but a slow stream enters the molten mass already in the mold more gently, stirs it less, and disturbs less the faster cooling of the lower part of the ingot because of its earlier entry into the mold. This distinction is of especial importance during the latter part of the pouring, when a slow stream may leave the lower part of the metal almost wholly unaffected.

This conclusion, which I reached by *a priori* reasoning, I find is borne out by experience in crucible-steel practice, in which rapid pouring increases the depth of the pipe; and Professor Stoughton and I found that narrow deep wax ingots, though they piped to within 10 per cent. of the bottom when poured very rapidly, yet formed only a very shallow pipe, reaching only to 13 per cent. of the length of the ingot, when poured very slowly.

§ 42. *Casting with the large end of the ingot up* (ingot *B* of Fig. 22) instead of down (ingot *A* of Fig. 22) clearly must increase the lag of the cooling of the upper part of the ingot, because the thicker end of the ingot cools more slowly than its thinner end, and must thus shorten the pipe and raise the segregation. In a direct test, in which steel from the same heat was cast into molds which were alike, except that the large end was up in one of them and down in the other, Mr. J. O. E. Trotz found, on

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A *B* *C*

FIG. 22.—CASTING WITH THE LARGE END UPPERMOST, AND WITH A SINKING-HEAD, SHORTENS THE PIPE IN STEEL INGOTS See §§ 42 and 44.

Ingots *A* and *B* were cast simultaneously through two nozzles from the same ladleful of steel of 0.50 per cent. of carbon, into molds 54 in. long, 6.75 in. square at the small end and 8.75 in. square at the large end, and in every respect alike, except that the large end was at the bottom in case of ingot *A* and at the top in case of ingot *B*. The pipe could be traced to a depth of 75 per cent. of the length of ingot *A*, or much further than is shown in this figure. Ingot *C* was cast with a sinking-head, a fire-brick sleeve around the contracted part at its top. (J. O. E. Trotz, Esq., private communication.)

breaking the resulting ingots across transversely, that the pipe stretched down less than 21 per cent. of the length of the ingot

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with its large end up, but more than 73 per cent. of that with its large end down. The molds were 64 in. long, 6.75 in. square at the small end, and 8.75 in. square at the other.

In another direct test he cast steel of 0.50 per cent. of carbon, from the same ladle, simultaneously through two nozzles into the ingots shown at *A* and *B* of Fig. 22, in molds which, as before, were alike except that the large end of one was up and that of the other down. The pipe, which in *B* is only rudimentary, in *A* could be traced to a depth of 75 per cent. of the length of the ingot. The illustration does not show the full

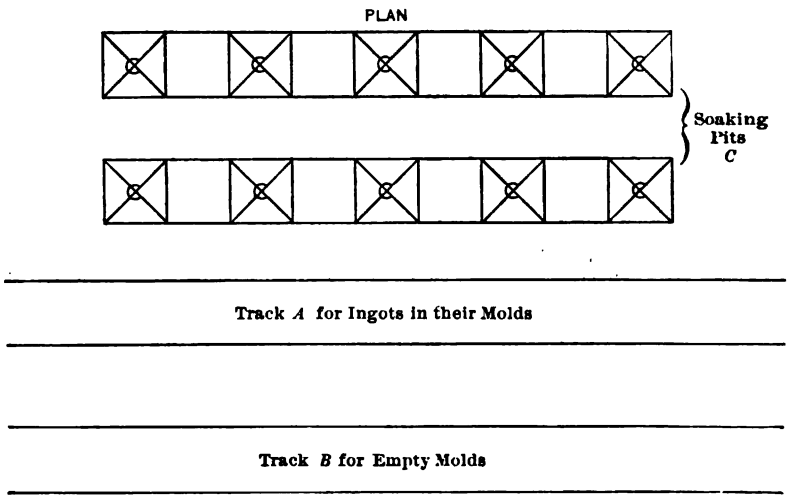

FIG. 23.—CAR CASTING, F. W. WOOD'S SYSTEM.

length of the pipe, because, when the ingot was split open, the path of rupture did not follow the pipe through the lower part of its length. The convex tops indicate that the casting-temperature was low. The molds were 54 in. long, 6.75 in. square at the small end, and 8.75 in. square at the other.

Again, Professor Stoughton and I found that strongly tapered wax ingots with the large end down piped very much deeper than exactly like ingots cast with the large end up.

§ 43. *Administrative aspect of casting with the large end up.* It is true that casting with the large end down has a certain advantage from an administrative point of view, because from an

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ingot so cast the mold can be removed more readily with the means now in use than from one of which the large end is uppermost. Indeed, our large American works have been planned for casting with the large end down. The ingots are cast in molds which stand upon cars, and, without waiting for the steel to solidify, a train of these cars is carried to track *A*, Fig. 23, beside the soaking-pits. Here the molds are lifted from the ingots and placed on another set of stools standing on cars on track *B*. The ingots which have thus been left bare, standing on their stools on the cars on track *A*, are next lifted

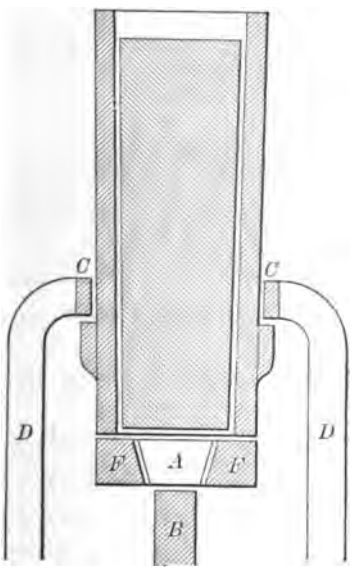

FIG. 24.—PROPOSED STRIPPING SYSTEM, MOVABLE STOOL-PLUG.

into the soaking-pits, *C*, *C*. Thus, a single moving of the molds and a single moving of the ingots puts the latter into the soaking-pits, and the former upon a fresh set of stools, ready, as soon as they are cool enough, to receive a fresh lot of steel.

But the advantage of this simplicity of administration may prove to be outweighed by the disadvantage which is inherent in this system, the disadvantage of making the pipe deeper-reaching and the segregation deeper-seated than they would be if the large end were uppermost.

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Two ways of stripping ingots cast with their large end uppermost have suggested themselves to me, and still better ones may occur to others. The first is to have a movable tapered stopper, *A*, conical or pyramidal, in the stool of each mold, as shown in Figs. 24 and 25. When the train of full molds has come beside the soaking-pits, the ingot is raised out of its mold by pushing up this conical piece by means of a ram, *B*, from below. The protruding ingot is then grasped by a pair of tongs from above and transferred into the soaking-pit, while the mold is left standing on its stool and on its car, ready to receive a fresh lot of steel as soon as it has cooled sufficiently. This way is, in one sense, even simpler than the present way indicated in Fig. 23, because the molds do not have to be

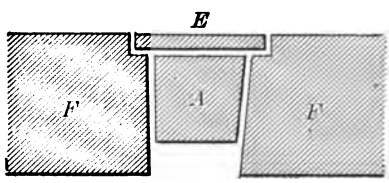

FIG. 25.—CAR CASTING, MOVABLE STOOL-PLUG, LARGE SCALE.

moved at all. There is only a single moving, that of the ingot.

If the molds were used too long or too hot, so that the ingot stuck to them instead of rising freely within them, they would have to be held down, for instance either by horizontal fixed bars, *C*, held by posts, *D*, or else by a piece projecting down from the crane which is to lift the ingot.

The natural objection which one raises to such a plan is that the steel may run down through the crevice between the stopper, *A*, and the rest of the stool. Though there is no clear reason why the steel should run through this crevice more readily than through that between mold and stool, yet if it did it would be likely to grip the stopper, *A*, so that it would not fall back with the plunger, *B*, when this was later lowered. Should it prove that this was really likely to happen, then a heavy plate-steel or even cast-iron cover, *E*, Fig. 25, could be set upon the stopper, *A*, with a layer of clay-wash between

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them, which ought to arrest any molten steel which worked into the crevice between *E* and *F*.[13]

The second way which I propose of stripping ingots cast with the large end up is sketched in Fig. 26. Here the full molds are lifted from their stools and set upon the stripping-stand, *G*. Then a plunger from beneath pushes each ingot up through its mold, which, meanwhile, is held down, either by a clamp or by a projection from the crane above. This crane carries tongs, which grasp the emerging ingot and transfer it at once to the soaking-pit. The empty mold is then returned to its stool on the car still standing beside it on track *A*.

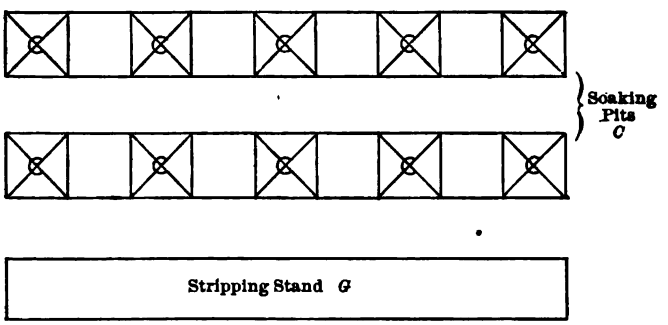

FIG. 26.—CAR CASTING, TWO MOVEMENTS FOR MOLD.

The disadvantage of this method is that it involves an additional moving for the mold. This has to be moved first from its car to the stripping-stand, and then from the stripping-stand back to its stool. But the cost of this additional moving may be outweighed by any material reduction in the needed length of cropping, or any material lessening of the segregation, brought about by having the large end uppermost.

§ 44. *The use of a sinking-head* or any other device for retard-

[13] When I wrote this, I thought that my invention was new. But, on inquiry, I find that Messrs. John A. Potter and William H. Morse have separately and independently invented this same contrivance. (See U. S. Patents, No. 601,083, March 22, 1898, and No. 735,795, August 11, 1903.)

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ing the cooling of the top of the ingot, and for feeding molten metal into the pipe as it forms, evidently should favor the lagging of the solidification of the ingot-top. Prominent among such methods is that of making the upper section of the mold, that part which contains the tapered top of the ingot, shown in Fig. 22, *C*, p. 61, of a fire-brick piece, a continuation of the usual cast-iron mold which serves to hold the rest of the ingot. This fire-brick prolongation is strongly pre-heated, and set in place just as the steel is to be cast in the mold. Other methods are to keep the ingot-top hot by means of a gas-flame (Riemer's process), a coke fire, or a large mass of molten slag poured into the upper part of the mold.

Closely related to this is the regular practice of many crucible-steel works in casting large ingots, and, indeed, all ingots large enough to need several cruciblefuls of steel. In this practice a few specially hot cruciblefuls are held in reserve in the furnace until the ingot-mold is nearly full, and then they are poured into it last, in order to raise the temperature of the top of the ingot relatively to the rest of it. If this practice is followed the pipe is short; whereas, if the cruciblefuls which are to form the top of the ingot are drawn from the furnace needlessly early and carelessly left standing on the floor, so that they cool off materially, the pipe may stretch down much deeper.[14]

The cooling of the bottom of the ingot is hastened spontaneously by the cooling effect of the thick cold cast-iron stool which forms the bottom of the mold. From the middle of the length of the ingot heat escapes only outwards; at its bottom heat escapes both outwards and downwards.

§ 45. *Segregation increases the lagging of the cooling of the upper part of the ingot.* The segregation which occurs in the freezing of a steel ingot is due to the fact that its freezing is selective—*i.e.*, that each layer of the molten steel in the act of freezing breaks up into two parts, a less fusible part which actually freezes, and a more fusible part which remains unfrozen, as part of the still molten central lake. Thus, in effect, each layer as it freezes ejects into this lake some of its more fusible matter, especially its carbon, phosphorus, and sulphur, and this

[14] William Metcalf, private communication, May 12, 1906.

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leads to a progressive concentration of these impurities in the molten lake and eventually in its last remaining drops, the last-freezing part of the ingot, which thus is the richest of the segregation. That is to say, at a given instant during freezing a single layer of nearly uniform composition should be deposited on the bottom and on all sides of the molten lake; and as segregation and the consequent enrichment of the molten lake take place progressively, and as the richer the lake the richer should be the layer which freezes out of it, so each layer deposited should be slightly richer than the preceding, and these successive layers should thus lie, onion-wise, around the last-frozen and most-segregated spot.

§ 45 A. *Lightness of the segregated matter.* Each layer as it starts to freeze along the shores and bottom of the molten lake may, for simplicity, be assumed to have the average composition of the molten lake which deposits it. But, as the part of this layer which does not freeze but is rejected and injected into the molten lake, is richer in carbon, phosphorus and sulphur than the average of the layer itself, it is richer also than the lake into which it is thus injected. Further, because the carbides, phosphides and sulphides of iron are lighter than iron itself, this injected matter should be lighter than the average of the lake into which it is injected. Thus there forms around the deep shores of the lake a layer lighter than the rest of the lake, and this should lead to a slow upward current around its shores, and a downward current about its axis, and thus these impurities should be concentrated upwards. But, as these impurities make the metal more fusible, this migration should make the upper part of the molten lake more fusible than its lower part, and this should retard the freezing of the upper part of the ingot.

There is another thing which may reinforce this tendency. If steel behaves like most substances and contracts as it cools towards the freezing-point, then the hottest of the molten metal would be the lightest, and the coolest would be the heaviest, and this natural difference in density would make the hottest metal rise towards the top and the coolest sink towards the bottom, thus further concentrating the heat at the top of the ingot, and thus further retarding the freezing of the top. In

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short, both the more impure and hence the lighter and the more fusible of the steel, and the hotter of the steel, should migrate towards the top, which would therefore freeze more slowly than the bottom, both because more fusible and because hotter.

Further, this upward concentration of the hotter and more fusible steel should be progressive, so that the imaginary central point about which the freezing takes place in concentric layers, instead of being fixed, should rise slowly as freezing itself proceeds.

§ 46. *The high position of the segregate tends to show that steel does not expand in cooling towards the freezing-point.* By reversing our process of reasoning, we find here evidence that molten steel does not, like water, expand as it nears the freezing-point. The fact that water thus expands, as shown in Fig. 27, explains easily why the segregate in ice ingots lies below the center, instead of above, as in the case of ingots of steel. To simplify our ideas, let us suppose that, in the central unfrozen lake of a freezing ice ingot, all the water was between 4° and 0°, or between *a* and *b* of Fig. 27. In this case the coldest of the water is the lightest and rises to the top, while the least cold is the heaviest and sinks to the bottom, and this hastens the cooling of the top and retards that of the bottom, and thus lowers the position of the last-freezing point or richest of the segregate. A further cause which makes in this same direction is that the impurities in the freezing water, which are rejected by the freezing layers and thus injected into the still unfrozen water, are chiefly mineral salts, which are both more fusible—*i.e.*, have a lower freezing-point—and heavier than the water, and therefore sink towards the bottom. These causes concentrate at the bottom of the freezing ice ingot both the warmer and the more fusible part, and thus in both ways retard the freezing of the bottom. Thus it is that I explain President Drown's observation[15] that the richest of the segregation in ice ingots is found below the center, a fact which seems to have puzzled even this acute reasoner.

Now, applying these facts to the case of freezing steel, if the steel expanded, as sketched in Fig. 10, p. 34, as it cools through

[15] *Journal of the New England Water Works Association*, vol. viii., p. 50 (1893).

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the last few degrees above the freezing-point, then the hottest of the molten steel should be the heaviest and should sink towards the bottom, while the least hot should be the lightest and should swim to the top of the molten lake; and this should retard the freezing of the lower part of the ingot and hasten that of the upper part, so that the last part to freeze would be below the center, or, in fine, that the richest of the segregate would be below the center.

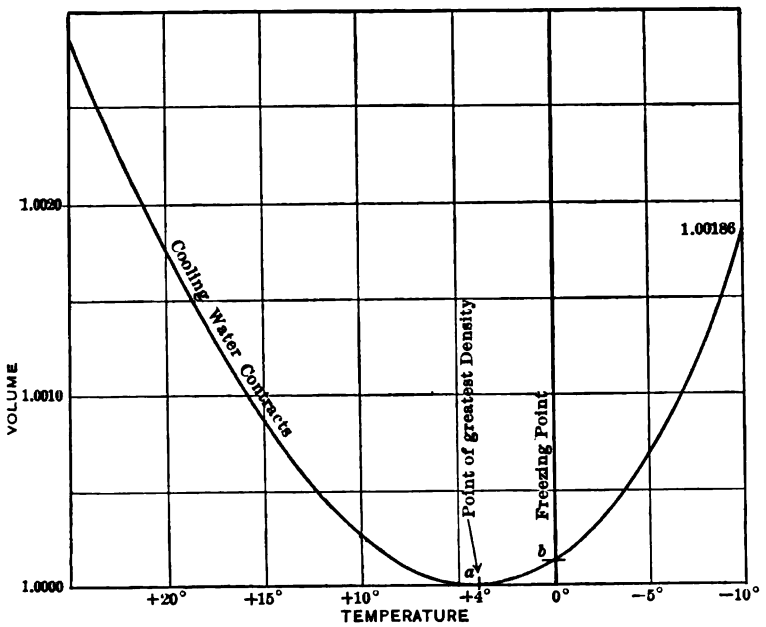

FIG. 27.—WATER EXPANDS IN COOLING FROM 4° C. DOWNWARDS.

But in point of fact the richest of the segregate is always found above the center, and generally very far above it, even when the other conditions of the case tend to bring it below the center. I refer to such conditions as having the larger end of the ingot below instead of above, and cooling the top of the ingot with water.

When this line of reasoning first occurred to me it seemed attractive. But on further examination it lacks cogency. For, on the hypothesis that cooling steel, like water, does expand in approaching the freezing-point, though it is true that the hottest of the steel would sink and the coolest rise, yet this might

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not prevent the rise of the impurities rejected by each successive freezing layer; so that there may be two simultaneous and opposite concentrations going on, that of the hotter and heavier steel downwards and of the more impure and lighter steel upwards. This might even go so far that the richest of the segregate should not be the last-freezing point; because as the impurities swim up and the hotter steel dives down, the richest of the segregate may lie well above the middle of the ingot, though the hottest point lies well below that middle; and the richest of the segregate may be so much cooler than the hottest point that it freezes first in spite of it being made more fusible by its very impurity.

But, though not conclusive, this fact that the richest of the segregate invariably lies well above the middle of the ingot is certainly strongly suggestive, and, as far as it goes, it strengthens the presumption against the hypothesis that molten steel expands as it nears the freezing-point. Of course, it throws no light whatsoever on the further question, whether steel expands in the very act of freezing and in cooling below the freezing-point.

§ 47. *Liquid compression.* That compressing the steel while it is solidifying—*i. e.*, while the pipe is forming—should tend to lessen this pipe by forcing the molten and pasty interior into it as it forms, or, indeed, after it has formed, needs here to be mentioned only. In §§ 60 to 71 we shall ask how liquid compression may be made most effective, both for raising the segregate and for closing the pipe.

§ 48. *The formation of blow-holes*, as we saw in § 24, lessens the pipe, and on this account the steel-maker habitually permits them to form, but in such small quantity and in such a position that they shall not harm the steel materially. Given a normally low casting-temperature, normal freedom from gas brought about by sufficiently boiling the metal after the last addition of ore, and a normal slag, the quantity and position of the blow-holes may be regulated by the additions of silicon, manganese, and aluminum made just before teeming, whether in the furnace or in the casting-ladle. These additions severally and jointly lessen the blow-holes; they should be added in such amount as to permit the formation of a small quantity of deep-seated blow-holes. For instance, under the special conditions

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of Brinell's well-known experiments, if the sum of the percentage of manganese plus 5.2 times the percentage of silicon is as much as 2.05 per cent., the steel will be so free from blow-holes that it will pipe badly. If this sum is reduced to 1.66 per cent., there will be just that small quantity of hardly visible blow-holes which will nearly efface the pipe. But if this sum is between 1.16 and 0.50 per cent., the blow-holes are so large as to be harmful, and they cannot be effaced by welding, because they lie so near the skin of the ingot that their walls are oxidized by the infiltering atmospheric oxygen, so that the contact of metal with metal, necessary to welding, is lacking. But, finally, if this sum is as small as 0.28 per cent., the blow-holes which form are so deep-seated as to be harmless, because their sides will not be oxidized, and therefore they will weld up completely in rolling, and will thus disappear.

If 0.0184 per cent. of aluminum is added, the effect is the same as if the sum of the percentage of manganese added plus 5.2 times that of the silicon were 1.66 per cent., so that here 0.011 per cent. of aluminum is the equivalent of 1.00 per cent. of this sum of Mn + 5.2 Si.[16]

§ 49. *The blow-hole forming period.* As when, in drawing from a soda-water siphon, we thereby reduce the pressure, so that the water, becoming supersaturated, evolves its excess of gas, and gas-bubbles form throughout the water and rise towards the surface; so, when the pressure within the ingot is reduced by the decrease in the extent to which the contraction of the outer walls outruns that of the inner walls, gas which has been dissolved by the great pressure in the already frozen but plastic inner walls is now evolved, and, unable to rise towards the surface, is yet able to push aside the surrounding steel sufficiently to coalesce into bubbles or blow-holes. Let us go on to consider this in more detail.

We have seen that during the early part of the pipeless period the contraction of the outer walls outruns that of the inner walls so greatly that each presses strongly against the other, but that this lagging of the inner walls gradually decreases, and

[16] *Journal of the Iron and Steel Institute*, vol. lxi. (1902, No. I.), pp. 333 to 353. "Iron, Steel and Other Alloys," pp. 368, 369.

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with it the pressure between inner and outer walls decreases, until finally this pressure becomes zero.

Indeed, it should turn from compression into tension. That is to say, the inner walls should henceforth be under slight tension, like the india-rubber which, in Fig. 4, p. 16, is pulled inwards by the strings. But any such tension is likely to be small, for an obvious reason. During the compression period, great compression can arise, at least in case the upper crust of the ingot freezes across firmly, because the inward pressure of the outer walls against the inner ones is resisted by the nearly incompressible molten lake against which the inner walls are compressed; but the outward drawing of the inner walls, after their rate of contraction has begun to outrun that of the outer walls, is not at all opposed by the molten metal within.

Suffice it for our present purpose to recognize clearly these two distinct periods: first, of strong compression, gradually diminishing to zero; second, of probably slight tension.

The evolution of gas depends upon the balance between existing pressure and existing solvent power for gas. If the metal contains more gas than suffices to saturate it for existing temperature and pressure—*i.e.*, if it is supersaturated—it normally evolves its excess of gas; if it contains less gas—*i.e.*, if it is not supersaturated—it does not normally evolve gas.

In general, the solvent power falls as the pressure falls; and in general it rises as the temperature falls. Thus, to heat a solid, for instance charcoal, may expel part of its dissolved gas; and a tumbler of water drawn cold from the faucet gradually evolves gas, as it stands and warms up on the sideboard. We are all familiar with the bubbles which form slowly on the sides of the glass under these conditions.[17]

[17] Certain metallurgists contend that the solubility of gases in molten steel decreases as the temperature falls towards the freezing-point, and that in this respect molten steel differs radically from other liquids in general. They point to their observation that, when in the open-hearth furnace the charge has ceased to boil, boiling may be induced by shutting off the supply of gas altogether, which no doubt lowers the temperature. They overlook a simultaneous effect of this same cause, that of increasing the strength of the oxidizing conditions to which the upper surface of the slag is exposed, by playing upon it a stream of white hot air now unmixed with gas. This may well act by increasing the proportion of ferric to ferrous oxide in the slag, and thereby inducing that slag to oxidize the carbon still remaining in the molten metal beneath, with evolution of carbonic oxide, which indeed is what the boil really consists in. In short, it is probably

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But though it is perfectly true that, as cooling proceeds, the solvent power for gas increases, yet when our cooling reaches the freezing-point this rise of solvent power abruptly turns into a rapid fall during the very act of freezing, again changing to a rise as the frozen metal cools from the freezing-point downwards.

If, to imagine an ideal case, steel cools, freezes, and then cools further, all at atmospheric pressure, as it cools from its casting-temperature towards the freezing-point the evolution of gas should slacken till the freezing-point is reached, should then become active during freezing, and should again slacken after freezing is complete, and the metal starts on its long journey of cooling from the freezing-point to the atmospheric temperature. It is true that we do not habitually see this slackening just before freezing sets in, in the steel in our molds, because as soon as we begin pouring into the mold some part of the steel actually begins to freeze, and therefore to evolve gas.

In short, if there were no change in pressure, gas should be evolved during freezing, but not while the steel is cooling from the freezing-point down.

Let us now consider how this course of events is affected by the changing pressure in our ingot, first a rise and later a fall.

The early rise of pressure, raising the solvent power of the freezing steel, lessens the quantity of gas which it evolves in freezing, and thus increases the quantity of gas stored up in the frozen steel. With the subsequent fall of pressure the solvent power, too, will fall, and in certain layers it is likely to fall by an amount which will so far exceed the simultaneous rise of solvent power, due to simultaneous fall of temperature, as to supersaturate these layers with gas, with the result that they will start to evolve the gas which they hold in excess of their present saturation-point. If those layers are so cool as to be rigid, this evolved gas must work its way outwards slowly as

through strengthening the oxidizing conditions, and not through lowering the temperature, that cutting off the gas induces a boil. We should be reluctant to assume that the common laws of nature are reversed, and should seek diligently for other explanations of any phenomena which, at the first superficial glance, seem to suggest such reversal. For examples of the great decrease of the solubility of many gases in water and other liquids as the temperature rises from 0° to or towards 100° C., see Landolt und Börnstein, *Physicalisch-chemische Tabellen*, p. 256 *et seq.* (1894).

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best it can. If they are so hot as to be soft and plastic, the evolving gas will coalesce into bubbles of some size; and, indeed, into each incipient bubble gas will evolve the more freely because the surface-tension of the bubble decreases progressively as its radius increases.

§ 50. *Normal position of blow-holes.* In the cross-section of a common steel ingot, Fig. 28, we note three distinct zones: an outer one, free from blow-holes; an intermediate one, containing a ring of blow-holes; and a central one, free from blow-holes. The outer clear ring and the intermediate ring of blow-holes are strikingly seen in our common ingots of artificial ice.

The intermediate ring represents already solid metal which, at the time when the pressure is falling, is (1) so rich in gas, and so hot, and therefore with so low a solvent power for gas, that the fall of pressure suffices to supersaturate it, so that gas evolves within it; and (2) so hot and soft that this gas can push it aside and form blow-holes.

FIG. 28.—NORMAL DEEP-SEATED BLOW-HOLES IN STEEL INGOTS. Brinell.

The outer ring represents metal either too cold and rigid, at the time when the pressure is falling, thus to permit blow-holes to form, or so cool, and therefore with so great solvent power for gas, that the fall in pressure fails to supersaturate it.

The inner circle may represent metal so poor in gas, because frozen under such slight pressure, that the fall of solvent power, due to later small loss of its small initial pressure, does not outrun the simultaneous gain in solvent power due to cooling, or does not outrun it enough to supersaturate the metal with gas. Hence no gas evolves within it, and no blow-holes form. Or it may represent metal so mobile that the gas evolved within it is able to work up and out into the still molten part, to swim through this to the top of the molten lake, and to work its way out through the holes, or at least pores, which are almost certainly always present in the top crust of the ingot.

The general shape of the blow-holes of this outer ring, as shown in Fig. 22, *A*, *B* and *C*, p. 61, and Fig. 11, p. 36, seems,

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on the whole, to agree better with this latter hypothesis. The side blow-holes are not horizontal, but tilted; the inner end of each blow-hole is higher than its outer end, as if it formed in more mobile metal, and thus had been able to rise slightly by gravity, though not enough to overcome surface-tension and free itself from the outer end of the blow-hole. The upper end of the blow-holes along the bottom of the ingot is usually much larger than the lower end, as if formed in more mobile metal, and in many cases the shape of the bubble suggests that it is the relic of a once longer bubble, of which the upper part has detached itself and risen to the surface. In ice ingots these bubbles are usually greatly elongated, and they curve gradually upwards as if, growing with the growth of the walls, the nascent inner end of the bubble had kept rising line by line as it lengthened, ever pressing upward against the not yet rigid wall, with whose growth its own keeps pace.

That the longer axis of each blow-hole should be normal to the nearest cooling surface, or, in other words, parallel with the axis of the columnar crystals between which it forms, is most natural. Each crystal, as it grows, would naturally reject into the space between itself and its neighbors any gas which it was compelled to expel.

My purpose in giving these general considerations about blow-holes is to stimulate others both to think about their causes and to publish the results of their own observations, with the further object of helping towards the formation of a true theory of their formation and prevention, both in order that the known facts may be conveniently and clearly grouped, and also that, by means of such a theory, we may predict laws not yet known, and thus develop further knowledge of direct value. At present I confess to great difficulty in forming any theory which is consistent with all of the facts, or even with nearly all of them. One which I prepared with great care for this paper I find so defective that I have suppressed it.

II. Segregation.

§ 51. *Precautions against segregation.* Before taking up these precautions, the reader should have clearly in his mind the picture of segregation drawn in §§ 45, 45A and 46, pp. 66 to 70.

To lessen the irregularity of composition which segregation

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causes, we should either restrain segregation or cut off and reject the segregate, or do both. If we are to cut it off, then we should aim to raise it—*i.e.*, to cause it to form as near the top of the ingot as possible—so as to reduce to a minimum the quantity of metal which thus has to be sacrificed.

§ 52. *In order to raise the segregate* we should retard the cooling of the upper part of the ingot, by the means described in §§ 39 to 44, pp. 58 to 66, so that the last part to solidify, which will be the richest of the segregate, shall lie as near as practicable to the top of the ingot.

These means were:[18]

(3) Top-casting;

(4) Slow-casting;

(5) Casting with the large end up, and

(6) Keeping the top hot by means of a sinking-head or other device.

The purpose immediately before us when we were there studying these devices was to shorten the pipe. But as each of these steps did this by making the cooling of the upper part of the ingot lag behind that of the lower part, and as increasing this lag must tend to raise the position of the last-freezing point or richest of the segregation, it is evident that each of these steps must tend also to raise the position of the segregate.

Two other steps which certainly shorten the pipe may raise the position of the segregate; these are:

(7) Permitting deep-seated blow-holes to form, by adjusting the quantity of silicon and manganese, or their equivalents, and

(8) Liquid compression.

In passing let us note that the steps which shorten and lessen the pipe by lessening the virtual expansion—viz.:

(1) Casting in wide instead of narrow ingots;

(2) Casting in sand- or clay-lined molds—

ought not, for any of these reasons which we have been discussing, to raise the position of the segregate, because they do not act through increasing the lag of the cooling of the upper part of the ingot. Indeed, by causing the solidification to take place slowly, these steps may prove to increase the degree of segregation.

[18] For the sake of uniformity I here retain the designation-numbers already given these several steps in the preceding parts of this paper.

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§ 53. *Restraining segregation.* Turning now from the means by which the segregate may be raised towards the ingot top, without necessarily changing the degree of segregation, let us next consider the means by which segregation may be lessened, so that the metal may be more nearly homogeneous. Prominent among the means which are either known or thought to lessen segregation are these:

(9) Quieting the steel—*i.e.*, suppressing the evolution of gas during solidification, by adding aluminum or its equivalent;

(10) Casting in small instead of in large ingots.

Hastening the solidification, not only by casting in small ingots, but also

(11) By casting at as low a temperature as practicable;

(12) By casting in thick-walled cold iron molds, instead of in sand or clay molds; and

(13) By casting slowly.

Before going on to consider these means in detail, let me at once point out that, though quieting wild steel certainly seems to be a most effective way of lessening segregation, yet the effect of ingot-size and of the rate of cooling is in dispute.

§ 54. *Quieting the steel.* Mr. B. Talbot[19] has certainly made out a very good *prima facie* case for the theory that quieting the steel by additions of aluminum lessens segregation, and Mr. Stead[20] has shown clearly that it ought to do this, because this quieting, by suppressing the evolution of gas and the violent upward convection-currents which this evolution sets up, tends to change the mode of solidification from the onion-type,[21] in which freezing proceeds by depositing a succession of smooth concentric layers of frozen metal along the narrowing shores and walls of the submerged molten lake and thus gradually sweeping the segregate axis-wards, to the "land-locking"[22] type, in which it proceeds by sending out long pine-tree crystals, the interlacing boughs of which mechanically land-lock much of the molten metal, and thus prevent the carbon, etc., ejected from each freezing layer in these land-locked harbors and ponds from working its way by diffusion towards the central axis of the ingot.

[19] *Journal of the Iron and Steel Institute*, vol. lxviii. (1905, No. II.), pp. 204 to 223.

[20] *Idem.*, pp. 224 to 228.

[21] *Iron, Steel, and Other Alloys*, the Author, p. 85.

[22] *Idem.*, p. 95.

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Clearly a given molecule of, say, sulphur, which is land-locked by the out-shooting crystals in the walls themselves, is thereby prevented from traveling centerwards and contributing to the central or axial segregate.

§ 55. *Influence of ingot-size.* Though it is the very general and in my opinion wholly natural belief that large ingots segregate much more than small ones, yet the remarks of such distinguished metallurgists as Messrs. H. H. Campbell and B. Talbot certainly make rather against than for that belief. A very large mass of data which I have analyzed tends so strongly to show that the prevalent belief is right—*i.e.*, that large ingots segregate more than small ones—that I adopt this belief provisionally. But this question is so important that I have undertaken further experiments and a systematic analysis of the data at hand. Postponing a full discussion till this work is done, I now offer some thoughts on this general subject.

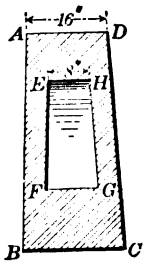

FIG. 29.—SEGREGATION SHOULD BE GREATER IN LARGE THAN IN SMALL INGOTS.

The influence of ingot-size, though it no doubt depends in part on the simultaneous influence of the rate of cooling, because large ingots naturally cool much more slowly than small ones, yet in addition depends on a second principle, which I will explain briefly.

At the time, T, when the freezing of a 16-inch ingot, *ABCD*, Fig. 29, has progressed so far that the volume of the molten lake remaining inside it is approximately that of an 8-inch ingot, *EFGH*, a great deal of segregation will already have taken place; the molten interior will now be much richer in the segregating elements than the already frozen walls. But from this time on the conditions in the further freezing of this molten lake will be pretty much the same as those in an 8-inch ingot, so that, at the end, in the 16-inch ingot there will be superadded to the segregation normal for an 8-inch ingot that which had already taken place at the time, T, when freezing had reached approximately the condition shown in Fig. 29. In short, there should be much more segregation in a large than in a small ingot.

Of course, this central 8-inch prism within our 16-inch ingot must cool very much more slowly than a common 8-inch ingot in contact with its initially cold mold. Suffice it here

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to point out, that there are here two distinct influences, the direct influence of ingot-size, and the influence of the rate of cooling. If, as certainly is to be expected, the direct effect of ingot-size is to favor segregation, then, if slow cooling itself increases segregation, large ingots should segregate more than small ones for a double reason. But if slow cooling opposes segregation, then the direct effect of ingot-size is opposed by the incidental effect of the slow cooling to which large size leads.

§ 56. *Influence of the rate of cooling on segregation. Important distinctions.* Let us at once recognize that we have here to do with three really distinct though closely related things:

(1) *The multiplication of phases*, as when a homogeneous liquid in solidifying yields not one but two or more products (*e.g.*, when molten cast-iron in solidifying yields austenite plus graphite or cementite or both); or when a solid solution is transformed into two or more products (*e.g.*, austenite transforming in the eutectoid range into ferrite and cementite);

(2) *Axial segregation*, the centerward concentration of the more fusible substances; and

(3) *Local coalescing* of the particles of the different phases, such as ferrite and cementite, into larger masses.

Let us further recognize that what we are now studying is the degree of axial segregation, and not the multiplication of phases or local coalescing; the concentration of the more fusible substances around the axis of the ingot, and not the degree to which graphite or cementite forms in solidifying, nor the completeness with which austenite passes into ferrite and cementite in cooling past the eutectoid range, nor yet the size which the particles of ferrite and cementite reach through the coalescing of the extremely minute particles which result initially from the transformation of the austenite.

Let us now consider the effect of rapidity of cooling on these three things.

§ 56 A. *The multiplication of phases* is certainly opposed by rapidity of cooling. When glass is cooled at the usual relatively rapid rate, it remains apparently a single solid solution; when it is cooled slowly enough it devitrifies, or splits up into distinct mineralogical entities and loses its transparency. So in the cooling of igneous rocks and slags. A slag which is simply

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a black glass when quenched in water, when cooled slowly splits up into different constituents or phases.

When molten cast-iron solidifies and tends to split up into solid austenite plus either the labile cementite or the stable graphite, it is extremely probable that the formation of the graphite plus the cementite taken jointly is restricted by sudden cooling.[23] It is further probable that a very rapid cooling may completely prevent the multiplication of phases here. Take, for instance, a low-carbon cast-iron with 2.25 per cent. of carbon, which on slow cooling should yield a mixture of 2 per cent. austenite plus enough graphite or cementite or both to represent the remaining 0.25 per cent. of carbon. It is probable that a sufficiently rapid cooling of such a cast-iron—for instance, by granulating it in a freezing mixture—would completely prevent the formation of either cementite or graphite in freezing, so that the cast-iron when it reached the eutectoid range would consist wholly of supersaturated austenite.

§ 56 B. *Local coalescing*, too, must evidently be lessened by rapid cooling, both directly and indirectly. That it is lessened is a matter of common observation. Coarse graphite in cast-iron and well-marked pearlite in steel are to be had only by slow cooling. An extremely rapid cooling prevents the formation of pearlite altogether, and gives us nothing but martensite or austenite or both. A slower cooling may give us sorbite, which we may conjecture to be nothing but extremely finely-divided pearlite; and usually, the slower the cooling the coarser is the pearlite—*i.e.*, the further has the coalescing of the particles of ferrite and cementite respectively progressed. (There are unexplained exceptions.)

Here rapid cooling probably acts both directly and indirectly; directly, by denying the time needed for the migration of the islets of ferrite and cementite which this coalescing implies; indirectly, by lessening the quantity of ferrite and cementite which form by the decomposition of the initial austenite. A cooling so rapid as to prevent altogether the formation of ferrite and cementite necessarily thereby prevents their coalescing. A cooling rapid enough to restrict the formation of ferrite and

[23] In exceptional cases it has been found that sudden cooling actually increased the proportion of graphite. These need further study. Compare Hogg, *Journal of the Iron and Steel Institute*, vol. xlvi. (1894, No. II.), p. 108.

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cementite to a very small amount, thereby restricts the quantity of these two substances which can coalesce. In other words, substances which do not exist because their formation has been prevented, cannot coalesce.

I understand that the segregation of lead-bearing statuary-bronze is of this class. A small quantity of lead is added to the bronze by certain founders, to make it softer and easier to cut with the finishing-tools. I understand that in order to prevent the spotting of the surface of these castings by the local segregation of the lead, they are stripped from their molds as soon as they are firm enough not to sag, and are cooled with water.

My purpose in dwelling on these things is to show that they differ essentially from the axial segregation which we are studying.[24]

§ 56 C. *Axial segregation.* It seems very clear that the direct effect of rapid cooling must be to lessen axial segregation for reasons like those which we have just been considering; but there is an indirect effect of rapid cooling which opposes this direct effect—viz., its diverting the course of solidification from the land-locking towards the onion type, or, in short, preventing the segregated matter from being locally pent up around the edges of the narrowing molten lake, and thus leaving it free to migrate centerwards. Let us consider these things separately.

First, let us note that axial segregation probably occurs almost wholly during the true solidification of the mass—*i.e.*, during the passage from the molten to the solid state—and is not materially increased by the transformations which occur in the eutectoid range, or by local coalescing. By the time that the eutectoid range is reached, the metal is so rigid that migration must be extremely slow; and axial segregation necessarily implies migration. The impurities found in the axial segregate must have traveled considerable distances in order to reach it. Local coalescing, instead of adding to axial segregation, should if anything work against it, as the least reflection shows. When two neighboring islets, for instance of ferrite, coalesce, the center of the resultant island is likely to be, not on the axis-ward but on the outer side of the center of

[24] See Appendix, p. 108.

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gravity of the two constituent islets, because that one of these islets which is initially nearer the axis of the ingot is slightly freer to move than the other, because it lies in slightly warmer and hence less rigid waters.

Let us now consider the conditions during solidification. To fix our ideas, let us consider the segregation of the carbon, for what is true of it is in a general way true of the other segregating impurities, the phosphorus and sulphur.

Let us picture in our minds that the mechanism of freezing is that a given layer in freezing splits up into two sub-layers of equal mass (note the distinction between layer and sub-layer), an impoverished one which freezes and an enriched one which stays molten; and that this latter immediately associates itself with its neighboring molten sub-layer to form a new molten layer, which will be the next to freeze by splitting up and depositing its impoverished half in like manner against the last frozen sub-layer, and turning over its other and enriched half to unite in turn with the adjacent molten sub-layer, and so on.

Then, to fix our ideas, let us consider the case of a 2.25 per cent. carbon cast-iron, Q', Fig. 30, of which the very first sub-layer has just deposited, with composition m, so that, because this sub-layer and the molten one just liberated are of equal mass, this latter has been enriched just as much as the frozen one has been impoverished, or by $m - n$, so that its composition is $m + m - n = p$.

The present composition of the layer which next will freeze, being made up in equal parts of this just liberated sub-layer with p per cent. carbon and a new sub-layer from the mother-metal with its old composition, m, has the composition $m + p \div 2 = p'$. But this littoral layer at once begins to grow poorer and poorer in carbon, because, being thus initially richer in carbon than the rest of the molten metal, its excess of carbon at once begins diffusing axis-wards into that mother-metal. Further, the slower the cooling is, the further will this axis-wards diffusion and the consequent impoverishment of the littoral layer have gone at the instant when it in turn splits up and deposits a second frozen sub-layer. Next, the further this impoverishment of the littoral layer has gone when it thus splits up, the poorer in carbon will be the new frozen sub-layer to which it gives birth, because the carbon-content of each

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sub-layer deposited is due to the carbon-content, not of the average of the whole molten mother-metal, but of the littoral layer from which this sub-layer is formed.

If, for instance, absolutely no time elapsed between the freezing of the first and that of the second sub-layer, then the composition of the layer from which the second sub-layer is born would be $p' = p''$, and the composition of the sub-layer freezing out of it would be m'; whereas, if diffusion had plenty of time to work, so that, by the time the second sub-layer froze the carbon-content of the layer from which it freezes had fallen back

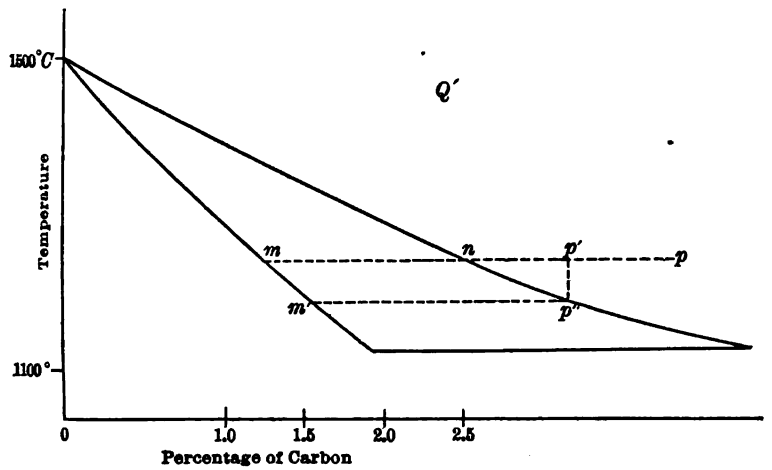

FIG. 30.—INFLUENCE OF THE RATE OF FREEZING ON THE COMPOSITION OF THE SOLID LAYERS DEPOSITED, AND THUS ON AXIAL ENRICHMENT.

very nearly to n, then the carbon-content of the sub-layer now freezing would be very nearly m.

Hence the more rapid the cooling the richer will be this second-freezing sub-layer, and the less will be the enrichment of the molten mass as a whole, and finally, the less will be that final axial enrichment which results from the enrichment of the successively frozen sub-layers.

To put this same thing in other words, each sub-layer as it freezes rejects a part of its carbon into the adjoining molten littoral layer. If this cooling is slow, so that some time elapses before this next layer itself freezes, then most of this rejected carbon will have worked its way centerwards by diffusion and

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convection, and will thus have swelled the axial enrichment in carbon. But if cooling is very rapid, then, when this next layer freezes, there will have been so little time for diffusion and convection to do their work of transporting this rejected carbon that most of it will still remain in this next layer when it freezes, so that an undue proportion of the whole will be locked up by this freezing, instead of migrating centerwards.

Each step in this reasoning seems to follow irresistibly from the preceding, so that the whole chain seems to demonstrate that the direct effect of rapidity of cooling must needs be to lessen axial segregation. And however much we may dispute the accuracy of the picture here drawn of the mechanism of solidification, yet whatever be our conception of this mechanism, this same conclusion must, I believe, follow from it.

There is a second and closely related way in which rapid cooling may lessen axial segregation. With a given carbon-content of the layer in the act of freezing, there is a normal and proper degree to which the sub-layer which actually freezes transfers its carbon to the sub-layer which remains molten. It is probable that rapidity of freezing interferes with this transfer, lessens the amount of carbon which is thus tranferred from the sub-layer which freezes to that which remains molten, and thus locks up an abnormal quantity of carbon in the frozen layers, and thus lessens the carbon available for axial enrichment.

To sum this up, rapid cooling probably acts in a double way: first, by preventing diffusion and convection from sending axiswards from the layer about to freeze its excess of carbon, so that that layer is unduly rich in carbon when it starts to freeze; second, by restricting the transfer of carbon from that fraction of this layer which actually freezes to that which remains molten, so that the freezing sub-layer locks up more than its due share of the carbon present. In short, with sudden cooling, the carbon in each layer as it starts to freeze is unduly great, and of this unduly large quantity an undue proportion is locked up in the sub-layer which freezes. Thus the axial enrichment is robbed in two ways.

§ 56 D. *How sudden cooling may increase axial segregation.* This it may do both by restricting diffusion in the frozen

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metal, and by diverting the solidification from the "land-locking" towards the "onion" type of freezing.

Though, as we have seen, diffusion and convection during freezing itself tend to increase the axial segregation by carrying the carbon ejected by the freezing metal away from the freezing region and towards the axis, yet after the metal has solidified, diffusion tends to lessen segregation, both axial and local, by re-distributing evenly the elements which segregation has localized. Hence, while slow cooling, by giving opportunity for diffusion, favors segregation during freezing, yet after freezing it tends to undo the segregation which has taken place.

That there should be less segregation when freezing is of the onion type than when it is of the land-locking type was explained at the beginning of § 54, p. 77. That slow cooling should favor the land-locking type, with its large, far-outshooting crystals, and that sudden cooling should oppose it, is both natural and in accordance with common observation. The steeper thermal gradient of sudden cooling leads to stronger convection-currents and a sharper evolution of gas, both of which tend to wash off the incipient pine-tree crystals. Again, the stronger convection-currents, by constantly changing the condition of the bath in which a given pine-tree crystal is growing, weaken the tendency of crystallization to adhere to its established axes, somewhat as continuous transplanting would kill a real pine-tree. Still again, the rapid forward motion of the shore-line in sudden cooling lessens the advantage which the established crystalline axes have of conducting heat outwards rapidly along well-established lines of thermal transit, in compact metal, and thereby lessens the attraction which their temperature thus lowered offers to the localizing of freezing—*i.e.*, to making it follow them, trunk and branch.

To sum up, we should expect rapid freezing to oppose axial segregation both by lessening the axis-wards diffusion (in the molten mother-metal) of the impurities rejected by the freezing layers, and by lessening the transfer of those impurities from each sub-layer as it freezes to its fellow sub-layer which remains molten. But we should expect it to increase axial segregation both by shortening the time which diffusion in the solid metal has for undoing the segregation which has occurred in freezing, and by preventing the land-locking of the impurities along

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the shores of the freezing lake, and thus leaving them free to migrate axis-wards.

Under these conditions we should expect that the balance of these opposing conditions would vary both in quantity and in sign from case to case; so that sudden cooling should sometimes increase and sometimes lessen axial segregation, and its effect should often be slight. There may be a certain intermediate rate which gives the least segregation.

§ 57. *The evidence briefly considered.* Turning now from deduction to induction, we find the contradictory evidence which we have thus been led to expect.

In the majority of cases which have come to my notice rapid cooling increases axial segregation, an effect exactly opposite that generally attributed to it. Thus Roberts-Austen[25] (then Mr. W. C. Roberts) found nine times as great axial segregation in a rapidly-cooled alloy of 92.5 per cent. of silver and 7.5 per cent. of copper as in a like ingot cooled slowly. In three out of four cases I found slightly more segregation of sulphur in cast-iron ingots cast in iron molds than in like ingots cast simultaneously in sand molds from the same ladle through a double funnel or distributer. These ingots were cast for me by Mr. T. D. West, Sharpsville, Pa., to whom my most sincere thanks are hereby given. Jars,[26] in 1781, asserted that pre-heating the molds lessened the axial segregation of certain copper-silver alloys, and E. Seyd proposed in 1871 the use of hot molds for casting gold and silver, because this made the bars "more equal in temper and in molecular arrangement."[27] This pre-heating would certainly lead to the slower freezing of the alloy.

Turning from these cases in which rapid cooling increases axial segregation, we have some in which it has the opposite effect. Thus I found a little more segregation in an extremely slowly frozen alloy of about 96 per cent. of copper and 4 per cent. of silver, than in this same alloy when cooled extremely rapidly.

[25] On the Liquation, Fusibility, and Density of Certain Alloys of Silver and Copper, by W. Chandler Roberts, *Proc. Royal Soc.*, vol. xxiii., pp. 481 to 495 (1875).

[26] "Je remarquai par des expériences que pour rendre les lingots d'une teneur plus égale dans toutes les parties il falloit que les lingotières fussent aussi chaudes qu'il est possible." Quoted by W. Chandler Roberts (*op. cit.*, p. 492) from Jars, *Voyages Métallurgiques*, iii., p. 270, 1781.

[27] Quoted by W. Chandler Roberts, *loc. cit.*

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The spot richest in silver had 4.88 per cent. of that metal in the slowly-cooled alloy, but only 4.57 per cent. in the other. These ingots were made under my direction by Dr. E. F. Kern and Mr. R. C. Blanchard, and the analyses were made by Dr. C. Offerhaus, whom I thank most warmly for their care and skill. The difference in degree of segregation is slight compared with the extreme difference in the rate of cooling. It is just such a moderate balance as might easily result from the struggle of opposing forces such as we have pictured.

Again, the relatively rapid freezing of the outer crust of a steel ingot seems to lessen segregation. Thus, in Mr. Stevenson's ingot, Fig. 13, if segregation had not thus been obstructed locally, there would have been an uninterrupted increase in carbon from the very outer crust to the central richest point in the segregate; but, instead, the carbon is higher, 0.60 and 0.61 per cent., in the lower corners than in any part of the interior except the immediate neighborhood of the pipe. This condition of affairs is not the exception but the rule, if we may judge from the considerable number of cases at hand.

This fact that there is a greater perceutage of carbon and the other segregating elements, phosphorus and sulphur, in the very first freezing parts than in those which freeze slightly later, that as we pass inwards these elements first decrease suddenly, and then again increase slowly, seems clearly to mean that the sudden freezing of the outer crust has locally obstructed segregation, whereas the much slower freezing of the deeper-seated parts has given segregation much freer play.

Finally, the evidence which I have as to the influence of the rate of cooling on the segregation in the freezing of aqueous solutions is self-contradictory. One eminent maker of very pure chemicals informs me that the purest crystals are to be had by rapid crystallization,[28] another of equal eminence asserts exactly the opposite.[29]

I infer that the effect of the rate of freezing cannot be marked and constant, because if it were it should have forced

[28] "A purer product can be obtained by rapid crystallization." "Rapid cooling with stirring to prevent the formation of large crystals always gives a more satisfactory product."

[29] "The rate at which crystals are made to deposit both from aqueous and from solutions of organic solvents, has a very material effect on the purity of the resultant crystals. The faster the crystallization, the more impure the crystals."

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itself on the attention of these competent observers, and it certainly should not allow them to form exactly opposite opinions.

The freezing of a metallic ingot and the deposition of crystals from an aqueous mother-liquor are of course strictly parallel, and the fact that in the latter case the mother-liquor is not frozen through and through does not affect the parallelism. Greater purity of crystals means a more thorough rejection of their impurities into the mother-liquor, and this in turn means a greater concentration of those impurities in the axial liquid when the freezing or crystallization nears completion.

The observation of one of these gentlemen that stirring increases the purity of crystals agrees with the reasoning in § 56 C. Stirring carries away from the surface on which freezing is taking place the sub-layer which, by the freezing-out of its mate, has just been enriched in the impurities present, and sweeps those impurities away into the general mass of the mother-liquor instead of leaving them in place to befoul the next deposited sub-layer.

In a later paper I hope to present the results of further investigations now in hand.

§ 58. *Means of varying the rate of solidification.* Of the means enumerated in § 53, the casting-temperature and the temperature and thermal conductivity of the molds need no explanation. It is evident that, if the steel is far above its freezing-point when cast, when it shall have cooled down to that freezing-point it will already have given up much heat to the walls of the mold, and thus in effect will have pre-heated that mold, which, because pre-heated, will abstract heat the more slowly from the freezing steel, and therefore the steel will from this time on cool more slowly than if it had been close to its freezing-point when cast, and thus had not so far pre-heated its mold when it reached the freezing-point.

But the effect of slow casting deserves a word of explanation, because here the conditions are not so simple. If casting is extremely slow, then, when the upper part of the ingot is pouring, the lower part will already have cooled far below its freezing-point, and hence heat will flow rapidly from the upper part into the now much cooler lower part. Hence the cooling and freezing of this upper part will be quicker than if the lower part had not thus been cooled off. Carrying this idea far

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enough, any given stratum of steel on reaching the steel previously in the mold will find itself underlaid by a thin stratum of steel, itself solidifying rapidly because of the coolness of the metal below, and hence quickly cooling the stratum now arriving to and past its freezing-range.

To go one step further, if each layer of steel solidifies before the next layer reaches it, then the matter axially segregated in a given layer cannot coalesce with that in the layer above, so that the vertical migration of the segregated matter would be completely stopped.

In pouring wax ingots extremely slowly into a mold surrounded by ice-water, Prof. Stoughton and I found that, instead of the single large axial segregate which we got at the same time by pouring half of this wax extremely rapidly into a like ingot, this slowly-poured ingot had a series of minute axial segregates, each nearly horizontal, as if one layer at a time had segregated independently of the metal above and below. This we referred to the freezing across of the surface at different depths when our pouring became unusually slow, or was even discontinued for a very brief time.

It is extremely probable that the degree of enrichment of these small axial segregates is much less than that of a single axial segregate, because this latter is enriched not only by the horizontal migration of matter at its own level, but besides this by the vertical migration of matter from above and from below towards the last-freezing spot. If the segregate is to be got rid of by boring out the axial part of the ingot, this slow-pouring procedure has much to recommend it. But if the segregate is to be removed by cropping off the top of the ingot, then this slow pouring may have the disadvantage of, in effect, lengthening out the segregate or at least lengthening out the region in which serious segregation exists, perhaps increasing the quantity of metal which has to be rejected on account of the segregate, and possibly even making it impracticable to get rid of the segregate by end-cropping.

§ 59. *May segregation be desirable in certain cases?* Segregation is itself a purifying process, concentrating the impurities into the last-freezing part. If enough of this can be cut away and rejected, an important degree of purification may be had. We have already considered removing the segregated part by

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cropping, but in many cases it is removed by boring out the central part; for instance, in preparing ingots for hollow forging, and in boring out the chambers of hollow projectiles. In still other cases the axial part, into which the impurities are concentrated by segregation, is relatively unimportant; for instance, because it is close to the neutral axis of a piece which has chiefly to resist transverse stress, or of a shaft which has chiefly to transmit rotary motion, or of an armor plate of which the face and back are the important parts.

In many such cases it may at first appear that segregation ought to be stimulated, so that its purifying effect may be increased. If segregation could be limited to the harmful elements, phosphorus and sulphur, this might indeed be a very attractive plan. Unfortunately, along with this purification goes, and must go, a corresponding irregularity in carbon-content.

This latter irregularity itself, and the harm which it does, naturally increase with the carbon-content of the piece as a whole; and hence, though in case of low-carbon steel it may be unimportant, and therefore to be tolerated because of the accompanying concentration of phosphorus and sulphur into a harmless position, yet in high-carbon steel its harm is likely to outweigh any such incidental advantage.

Indeed, each case must be judged on its own merits, weighing the harm which the irregular distribution of carbon may do against the good which may come from the concentration of phosphorus and sulphur.

In case of basic open-hearth steel, purity can in general be bought more cheaply by eliminating phosphorus and sulphur than by segregating them into a harmless position, and this should be especially true if the electric purifying processes keep their promise.

And though, in case of acid open-hearth steel, it may often be well to weigh carefully the penalty of irregular carbon-content, which we have to pay if we get our purity through segregation, against the high price of extremely pure raw materials which we should otherwise need; yet this latter price will probably be found less than that penalty in the great majority of cases, and, indeed, in a majority which will increase as time goes on.

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§ 60. *Fluid compression may raise the segregate* towards the top of the ingot, and so lessen the proportion of the upper part which must be rejected in order to get rid of the segregate. In ingot *D* of Fig. 1, p. 6, the richest of the segregation should lie about *K*, somewhat above the center of the molten lake, because the segregated matter is lighter than the rest, as pointed out in § 45A (p. 67). If, when matters have reached this stage, the ingot is compressed, the still-molten metal may be forced up so as to fill the empty space which now constitutes the pipe. In effect, we make use of the empty space, the pipe itself, as a receptacle into which we may squeeze the molten segregate, now lying below it.

The effectiveness with which we may thus raise the segregate into the pipe, into this space left vacant by the ebb of the tide, depends both on the time and on the manner of applying the compression. This should be applied late during the solidification, so that there may be a large cavity into which the segregate, now reduced to a small bulk, may be effectively lifted; and it should be applied lower down than the bottom of the pipe, so as to leave the pipe of its full size, and therefore with the maximum capacity for receiving the segregate. So far as I know, these considerations, which I will now elaborate, are here set forth for the first time.

§ 61. *Time of applying the compression.* If the compression is applied in stage *B* of Fig. 1, when the quantity of molten steel is still great, the quantity lifted would be great, and the distance through which it would be lifted would be small. If, on the other hand, compression is deferred until stage *D*, the volume of the pipe is greater and the quantity of metal to be lifted much smaller; so that in this case the vertical travel, or the distance which the segregate is lifted, is much greater. If the large end of the ingot is uppermost, and if the cooling of the top is retarded by the several devices of top-pouring, slow-pouring, and using a pre-heated sinking-head, and by the natural action of the segregation itself (§ 45), so that the state of affairs is more like that sketched in Fig. 32, then compression should have a very important effect in lifting the segregate.

Clearly, the later in the freezing the compression is applied, the farther will it lift the segregate towards the ingot-top, but the smaller will be the quantity of this segregate thus lifted,

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and the richer in impurities will be those axial parts of the ingot left after the segregate has thus been lifted. The reason for this last fact is that, other things being equal, the richer the mother-metal is in any given impurity, the richer in that impurity will be the steel which freezes out, layer by layer, from that mother-metal; so that as freezing and segregation proceed, and the mother-metal grows progressively richer in the impurities, so is each layer of solid steel which freezes out of that mother-metal richer in those impurities than the last preceding layer; and this goes on until the layers which are depositing may become prohibitorily impure.

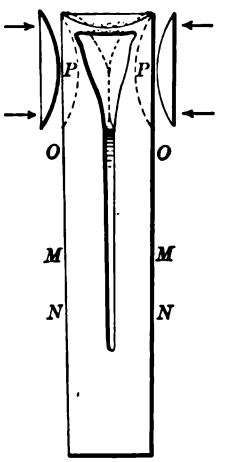

FIG. 31.—HIGH-LEVEL LIQUID COMPRESSION PREVENTS RAISING THE SEGREGATE.

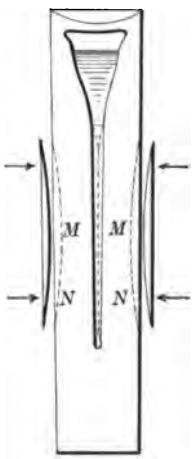

FIG. 32.—LOW-LEVEL LIQUID COMPRESSION RAISES THE SEGREGATE INTO THE PIPE.

The dotted lines show the state of things after applying the compression.

In view of this, the time at which we should aim to apply the compression should be that at which the percentage of impurities—for instance, of phosphorus—in the layers now freezing, has risen as near as is safe to the permissible limit for phosphorus. Compression applied then should leave in the unlifted frozen part no layers prohibitorily rich in impurities; and it should lift the segregate towards the top of the ingot, and thereby diminish the percentage of cropping needed to remove that segregate, as far as is compatible with having no part of the remaining ingot prohibitorily rich in any impurity.

This time could never be hit with exact accuracy; but careful experiment thus directed might enable us to time the com-

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pression advantageously. And, of course, the compression should not be so long delayed that the ingot's crust has become too thick and rigid to be compressed effectively by the means at hand.

§ 62. *The manner of applying the compression.* In order to lift the segregate most effectively, we should avoid narrowing the pipe itself, and we should chiefly narrow the molten lake, as, for instance, by applying pressure at points below its surface, as at *MM* and *NN*, Fig. 32. Thus, if we should first apply pressure at *PP*, Fig. 31, by pressing together a pair of convex pieces there, we should close up the cavity; and if, while holding it thus closed, we should next apply pressure at *OO*, *MM* and *NN*, this pressure could not lift the segregate, because there would no longer be an empty space into which to lift it. If, on the other hand, pressure is applied through like convex pieces at *MM*, Fig. 32, and none is applied at *PP*, then the compression should lift the segregate very effectively, because the volume of the empty space which is to receive it has not been lessened by the compression.

There are four prominent methods of applying the pressure: lengthwise (Whitworth's); sidewise uniformly (Illingworth's); endwise in a conical mold, which results in uniform sidewise compression (Harmet's); and sidewise, chiefly near the middle of the ingot's length (Williams's).

To refresh the reader's memory, I will first describe these processes briefly, and then consider how they compare as means of lifting the segregate, and thus lessening the cropping needed.

§ 63. *In Whitworth's system*[30] the ingot is cast in a vertical cylindrical iron mold, *L*, Fig. 33, strongly hooped, *K*, and lined with molding-sand, *N;* and it is compressed lengthwise by being pressed up against the fixed ram, *G*, during and after solidification.

§ 64. *In Illingworth's system*,[31] shown in plan in Fig. 34 at two stages, the ingot is cast in a vertical mold of the usual shape, but split lengthwise. During the casting of the ingot, the two halves of the mold are held a little apart, as shown at

[30] *The Metallurgy of Steel*, H. M. Howe, p. 155.

[31] Piping in Steel Ingots, by N. Lilienberg, *Trans.*, xxxvii., 238 to 247 (1907).

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I., and the crevice thus formed between them is temporarily stopped with specially shaped distance-bars. As soon as the crust of the ingot has solidified to a proper thickness, these distance-bars are pulled lengthwise out from between the two halves of the mold, and these halves are then forced together

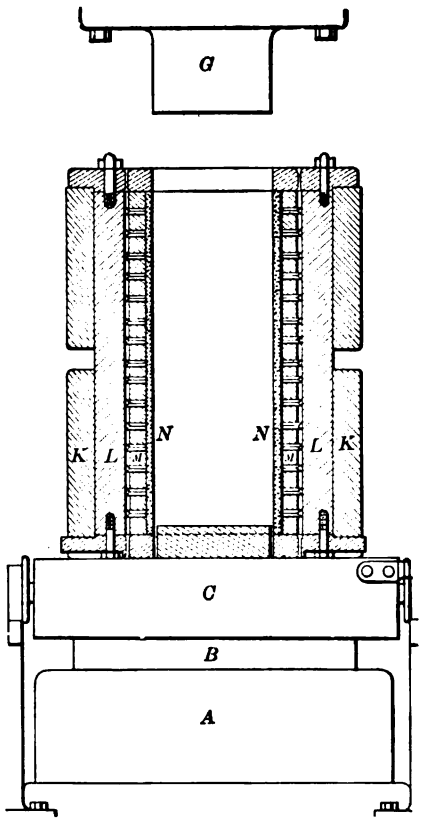

FIG. 33.—WHITWORTH'S HYDRAULIC PRESS FOR THE COMPRESSION OF STEEL INGOTS WHILE SOLIDIFYING.

A, main compression-cylinder. *B*, its plunger. *C*, the carriage on which the mold or flask sits. *G*, boss against which the steel in the mold is forced. *KK*, steel jackets for the mold. *LL*, the mold proper. *MM*, perforated cast-iron lagging. *NN*, inner sand lining.

by the ram, *F*, to the position shown at II., of course compressing the ingot horizontally by the amount by which the two halves originally gaped apart in I.

We should expect that the compression would force the steel out into the groove between the two halves of the mold, left

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vacant by the removal of the distance-bars; but we are told that it does not. If trouble from this source should arise, the distance-bars could be shaped as I have shown them, so as to make an initial groove on the side of the ingot, which the compression and the subsequent rolling would efface.

§ 65. *In the Harmet or "draft-compression" system*,[32] Fig. 35, the ingot, *AA*, is cast as the frustum of a slightly tapering cone, in a conical mold, and is then forced up towards the apex of the cone by means of pressure applied at its base. As when a tapered plug is driven into a tapered hole, the pressure exerted against the sides of the hole is enormous; so here,

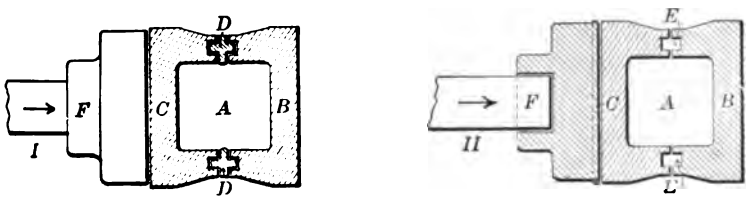

FIG. 34.—ILLINGWORTH'S PRESS FOR COMPRESSING STEEL INGOTS HORIZONTALLY WHILE SOLIDIFYING. Sectional Plans.

In I. the mold is shown ready for receiving the molten steel. Two distance-bars, *DD*, are set between the halves of the split mold, *B* and *C*. After the steel has been poured into the mold, these distance-bars are pulled out lengthwise, and the two halves of the mold are then forced towards each other by means of the ram, *F*, as shown in II. The convex edges of the distance-bars are for the purpose of making an initial depression in the side of the ingot, lest part of its side should be forced out as a fin or welt into the crevice between the two halves of the mold. N. Lilienberg, Piping in Steel Ingots, *Trans.*, xxxvii., 239 to 247.

because action and reaction are necessarily equal and opposite, there is, in effect, an enormous inward radial pressure upon the surface of the ingot. A moderate pressure at its base causes an enormous pressure on its sides. The progressive narrowing of the ingot as it travels up through the mold causes incipient but instantly effaced puckering, and forces the metal centripetally to fill up the pipe as fast as it forms.

§ 66. *In S. T. Williams's system*[33] the ingot is cast in a split mold, as shown at I. in Fig. 36. As soon as its walls have be-

[32] *Journal of the Iron and Steel Institute*, vol. lxii. (1902, No. II.), pp. 146 to 207. H. M. Howe, in the Report of the Commissioner General of the United States to the International Universal Exposition, Paris, 1900, vol. v. *Iron, Steel and Other Alloys*, H. M. Howe, p. 373. *Journal U. S. Artillery*, March–April, 1905.

[33] *Metallurgy of Steel*, H. M. Howe, p. 156; U. S. Patent 331,856, December 8, 1885.

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come firm enough to stand unsupported, the two halves of the mold are drawn apart, as shown at II., a liner, *B*, is slipped between ingot and mold, and the mold itself is covered with a strong cap. The right-hand side of the mold is then pressed to the left by the ram, *C*, forcing the liner, *B*, against the ingot, and bringing matters to the condition shown at III. The forcing in of the initial bump or convexity of the right-hand side of the ingot squeezes the molten steel up into the pipe.

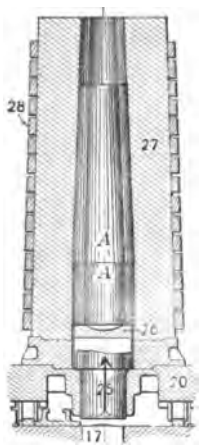

FIG. 35.—HARMET'S LIQUID COMPRESSION BY WIRE-DRAWING.

The ingot, *AA*, is cast in a strong conical mold, 27, reinforced with hoops, 28. Strong pressure at the base of the ingot, 26, forces it lengthwise of the mold, thus compressing it radially.

I should have expected some difficulty from the squeezing of the steel into the crevice between the two halves of the mold. But, when I saw the process in actual use, this trouble did not appear to arise, probably because, by the time the compression was applied, the bottom of the ingot had grown cool and firm enough not to squeeze into the crevice.

§ 67. *Relative effect of these four systems in raising the segregate.* Let us now consider the relative merits of these four systems, regarded as means of lifting the segregate, and thus lessening the quantity of metal which must be cropped from the top of the ingot in order to get rid of this segregate.

§ 68. *Whitworth's system.* As the outer walls of the ingot solidify and begin to contract, they tend to draw inwards and away from the walls of the mold, leaving an annular space between; but the lengthwise pressure, shortening those walls, and thereby thickening them, both squeezes them out so as to keep them in actual contact with the walls of the mold, and thickens them inwards as fast as the pipe-forming tendency gives any room for this inward forcing. If a pipe were allowed to form before the pressure was applied, then this inward forcing of the walls would in effect lift the molten interior into that pipe. And if the pressure is applied continuously throughout the period when the pipe tends to form, it in effect lifts this molten interior to fill up this nascent pipe as fast as it tends to

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form, so that at first it seems to offer a good means of raising the segregate, and thus of lessening the amount which has to be cropped off in order to get rid of the segregate. But there are three things which seriously interfere with the effectiveness of this raising of the segregate, as I will now explain.

The first is, that the compression tends to close up the pipe by squeezing its walls in earlier than it squeezes together the walls of the lower part of the ingot, because the first applied pressure will, of course, take its greatest effect where the walls of the ingot are the softest and thinnest, and this will be near the top where the pipe is beginning to form. The reason for this is, that the solidification at the bottom of the ingot outruns

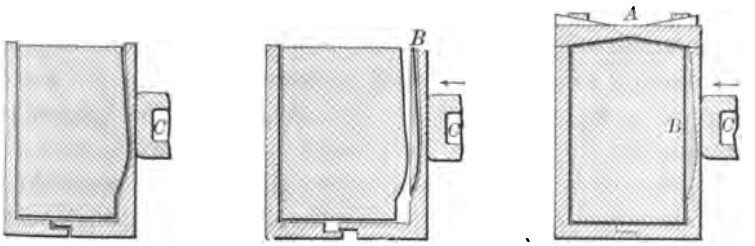

FIG. 36.—S. T. WILLIAMS'S ABDOMINAL LIQUID COMPRESSION FOR SOLIDIFYING STEEL INGOTS.

The ingot is cast in its mold, as shown at I. After its outer crust has solidified, the mold is opened, as shown at II, and a liner, *B*, is slipped between mold and ingot. A strong cap, *A*, is then fastened down, and by means of pressure applied through the ram, *C*, the abdominal protuberance on the ingot is forced in, so as to close the pipe and lift the segregate into it, as shown at III.

that at the top; first, because the bottom is cast first and is cooling off while the top is receiving fresh additions of hot steel from the casting-ladle; and, second, because of the cooling-effect of the bottom of the mold. In short, because at any given moment the upper part of the walls is thinner and softer than the lower part, the compression tends to close up the pipe faster than it squeezes up the metal from below to enter it; and this is our first reason why Whitworth's system is at a disadvantage as regards lifting the segregate.

The second reason is the fact that, in order to compress the ingot lengthwise, Whitworth's system has to compress lengthwise the firmest part of the ingot, to shorten the hollow column of solid steel which at any moment during the freezing

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has already solidified against the walls of the mold, and to shorten it by a direct pressure applied at its end, where it offers the greatest resistance to compression. In Harmet's system the pressure has the very great advantage of the wedge shape of the mold; in Illingworth's system the ingot naturally bulges in or crumples at its sides, along the plane where the distance-bars have been, where its walls are much thinner than they are at the corners. In Williams's system the flat side of the ingot at its thinnest part is forced in laterally. Compared with Whitworth's system, this is like attacking a column by horizontal pressure applied at the middle of its length, its part of least resistance, instead of by vertical pressure at its end, its part of greatest resistance.

Now, this very fact that Whitworth's compression attacks the ingot along its line of greatest resistance leads to the need of beginning the compression very early, before the column has become too strong to be compressed. But, as we have seen in § 61, this early compression, before the pipe has reached any considerable size, implies raising the segregate through only a very small distance.

The third reason is that Whitworth's endwise compression, in bulging out the sides of the ingot as fast as they tend to draw away from the walls of the mold, is like pressure at the end of an ill-hooped barrel, which makes its staves yawn open. Whitworth's compression, in like manner, is very liable to crack the thin and mushy walls of the ingot, which have neither the mobility of the liquid state nor the ductility of the solid state, to make them yawn open like the staves of a barrel, to squeeze the segregate out into them instead of raising it up into the pipe, and thus to give a series of hard longitudinal strips of segregated matter at the skin of the ingot.

This defect of Whitworth's system, that it does not lift the segregate efficiently, is quite apart from its defect of attacking the ingot along its line of greatest resistance, and thus of needing the maximum of power.

§ 69. *In Illingworth's process*, the amount by which the walls are forced together by the compression should be the same in the upper part of the ingot as in its lower part.

§ 70. *In Harmet's process*, although it is true that the pressure is applied at the base of a conical ingot, yet this results in

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a very powerful lateral compression, quite as when we drive a tapered plug into a tapered hole; and this lateral compression tends to raise the molten segregate up into the pipe, quite as in Illingworth's method. In one process as in the other, the amount by which the walls of the ingot are forced together should be the same in the upper part of the ingot as in the lower part.

§ 71. *In Williams's process*, Fig. 36, most of the lateral compression is in the middle and lower part of the length of the ingot, and the sides of the pipe are forced towards each other only relatively little; so that the chief effect of the compression, from our present standpoint, is to raise the segregate, and the amount by which it narrows the pipe and thereby impedes the raising of the segregate is very small. I do not know whether the inventor understood that his process might have this effect of raising the segregate. His object was to close up the pipe and blow-holes; but if the top itself is to be cut off and rejected after the segregate has thus been lifted into it, then it ought to be left uncompressed, so that the cavity in it which is to receive the segregate forced up from below should be left as large as practicable. In short, the compression should be restricted to the main body of the ingot, and the part which is to be cut off and rejected should not be compressed.

§ 72. *Summary.*—To sum this up, the beneficial lifting effect on the segregate should be the greatest in Williams's system, which compresses the ingot chiefly in the middle of its length; it should be the least in Whitworth's system, which compresses the ingot more at its top than elsewhere; and it should be intermediate in the systems of Illingworth and Harmet, which compress the ingot equally in all parts of its length.

III. Engineering Specifications.

§ 73. *Precautions in engineering specifications as to piping and segregation.* What steps should the buyer's engineer take to assure himself that the steel which he receives is free from injurious piping and segregation? Should he content himself with inspecting the finished products sent him, and not attempt to interfere with the manufacture, looking to the manufacturer for results, without question as to the means by which they are reached? Or, going to the other extreme, should he insist not

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only on specifying all the conditions of manufacture, but also on inspecting them all, so as to assure himself that his specifications are actually obeyed? As usual, it is the part of wisdom to go to neither extreme, but, relying rather on skill than on brute force, to plan the specifications and inspection so that the evidence which they give as to the fitness of the steel may be as conclusive as is compatible with due regard to the maker's interests; that it may be easy to get and yet convincing.

It is unwise to be content with inspecting the finished product. This, indeed, might do well enough if each heat of steel were uniform throughout; but piping and segregation, two of the worst defects, are local, and not even thorough inspection of the finished steel can tell where they are at their worst. Even if inspection of fifty places taken at random, and therefore in the dark, should show no harmful segregation, yet at a fifty-first the segregation might be intolerable; and this intolerable segregation might thus escape the most searching of inspectors if he were thus blindfolded.

It is unwise to go to the opposite extreme and insist on specifying and inspecting all the details of the manufacture, chiefly because this would probably have the effect of preventing the most skillful makers from bidding, and thus not only limit competition and facilitate collusive bidding, but also cut off your best sources of supply. It would prevent many skillful makers from bidding, because it would threaten to reveal secrets, the exclusive possession of which may be of the greatest value to the manufacturer.

§ 74. *Secrecy.* At first, those whose training is chiefly in civil or mechanical engineering may ridicule this, and talk about keeping information in at the cost of keeping information out, and of the short-sighted policy of secrecy and concealment in general. But there is a real difficulty here, which often forces the metallurgist into a secrecy not only inconvenient and galling, but expensive and dangerous, dangerous because secret processes, unprotected by patents, are a temptation to larceny and blackmail difficult to prevent, resist, or punish.

§ 75. *Uncertainty of patent-protection.* The fact that, in spite of this danger, men otherwise sagacious and conservative will pay very large sums for secret processes, is good evidence that this difficulty of which I speak is real and serious. It lies in

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the uncertain and insufficient protection which patents afford to many metallurgical processes, as distinguished from products. It is difficult to frame the patent specification, and extremely difficult to detect and prove infringements. Much of the value of the secret knowledge may lie in skillful execution of details, in the combination of steps each of which may be unpatentable because already known, just as no part of a sewing-machine may be patentable, yet the machine itself may be clearly patentable.

§ 76. *Difficulty of proving invention.* In case of such a mechanism we often can have some confidence that the courts will uphold a patent, because here the existence of real invention can readily be made clear. This is because all men are necessarily mechanics, dealing with statics and dynamics from the moment when their hands first grasp their infantile feet. Every motion which I take or which I see any man, animal, or other object take, instructs me in the essentials of mechanics. With chemical matters it is very different. They are more occult; they are not the object of daily thought and experiment. Still more so are metallurgical matters, based on the fearfully complex conditions and principles of physical chemistry, further complicated by the great ranges of temperature covered, each with its potent influence.

You hesitate to patent your metallurgical invention. To patent it reveals it to all your competitors with perfect certainty; "The Gods themselves can not recall their gifts;" but whether the courts will uphold your patent is extremely doubtful. Can you persuade the court that you have really exercised the inventive faculty? If, with all your skill, you fail to explain your metallurgical conceptions so as to force the unwilling controversial specialist to concede the truth of metallurgical views which to you seem absolutely unquestionable, how can you have any confidence that you can make a court, which has never thought of such questions, even understand you; and, granted such understanding, how can you hope to prevent it from becoming hopelessly confused by a skillful opponent?

§ 77. *Difficulty of detecting infringements.* Unpromising as the patentee's prospects thus are as regards proving that his invention really is an invention, they are equally unpromising as regards detecting infringements. If a machine or a mechanical

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appliance is patented, each infringement is open to detection. Each piece which infringes the patent is its own evidence of infringement as soon as it reaches the market, and it remains valid evidence as long as it exists.

But a metallurgical process is generally only a means towards a desired specific result, and to prove that this means has actually been used is likely to be very difficult; first, because we can rarely either know or prove that its specific result has been attained; and, second, because, even if we knew that it had, we could rarely know or prove that it had been attained by our means. If, for example, the process aims to give certain particular excellent qualities to the steel, you can rarely know that any particular lot of steel, or even that the regular output of a given maker, really has those particular excellent qualities; and if you did know it, it would be difficult to assure yourself that it was by the use of your particular process that these qualities had been given. If my secret dog-mixture is the best, the fact that other dogs have won the blue ribbon is poor proof that they have fed on my mixture. Not every thin man eats my "anti-fat."

To prove suspected infringement often requires litigation, which is very uncertain in its outcome, but certain to be very costly and likely to reveal any secrets which the patentee of the particular process in dispute may not have patented. Thus the protection of patented processes by litigation may easily mean the revealing of any unpatented ones; so that, unless the manufacturer is willing to open everything, he may reasonably be loath to open anything to the prying of the patent-attorney and expert.

In many cases infringement can be proved only by putting the suspect's workmen on the witness-stand, not an attractive procedure to those who would live in good-will with their neighbors, and one which, by provoking retaliation, may lead to exposing any secrets which you yourself may have.

In many cases the product can in the very nature of things give absolutely no token of having been made by a certain process. For instance, how can one know from examining a piece of steel in what particular way it has been recarburized, what the procedure in the open-hearth furnace has been, or how the temperature of the Bessemer converter in which it has been made has been governed?

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Even in case of clearly unpatentable inventions, reinventions of processes already disclosed, and therefore unpatentable, or processes covered by patents which have now expired, the manufacturer's exclusive knowledge may be of great and legitimate value. Of such a process he may find a valuable but apparently unpatentable application. He is under no obligation to inform his competitors that his quicker intelligence has discovered a useful way of applying this process, which to others has seemed useless.

For these and like reasons, to protect processes by patents is much harder than to protect products; and it is rarely wise to insist that all the steps of metallurgical manufacture shall be open to the inspection of the buyer's engineer, for this may result in excluding the most desirable bidders, and perhaps the only really competent ones.

We therefore ask, which steps of manufacture are most important to inspect, and which of these can and ought the manufacturer permit you to inspect? Of which is it to the buyer's interest to ask inspection, and of which can the manufacturer reasonably be expected to permit inspection?

§ 78. *Inspection at the rolls and shears, with axial drilling for the segregate*, seem the steps which are both the most searching and convenient for the buyer and the least objectionable to the maker.[34] A glance at the ingot before it enters the rolls shows its size, whether it has been top- or bottom-cast, and whether it has been cast with the large end up or down. The inspector further sees the finishing-temperature; he watches the cropping to see whether this goes properly beyond all unsoundness; and, probably most important of all, though hitherto overlooked, he can mark those blooms or billets which come from the upper end of each ingot, and mark their upper ends distinctively, so that he can later identify them, and by analyzing drillings from the axis of these pieces, learn whether all harmfully segregated parts have been cut off and rejected.

It is in the axis of the billet or bloom that the freezing must end, and therefore that the richest of the segregate must lie; and not only in the axis but in the axis near the upper end of the ingot, or at least near the upper end of the pipeless part of the ingot. If the conditions of casting and cooling are such

[34] Inspection of the finished product is of course assumed.

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that serious segregation occurs, and if the cropping is carried far enough to remove all piped parts, then the richest of the remaining segregate will lie in the axis near the upper end of the remainder of the ingots. And if the drillings are taken from the axis of each end of the bloom coming from the upper end of the remainder of the ingot, or from the axis of each end of the two blooms or billets nearest the upper end, then if any harmful segregation has been left uncropped and unrejected, its presence is almost certain to be detected in the blooms from at least one and probably in those from several of the ingots of a given heat.

The inspector who proceeds thus is unblindfolded. Light is given him as to the position of the segregate, so that his investigation of its extent and harmfulness is made intelligently and with knowledge. He probably will not, indeed, find the very richest of the segregate, for this richest spot is at a mathematical point which is far more likely to lie between the two ends of the billet or bloom which contains it than at the very end, where it could be found by drilling. But the concentration or segregation is grouped onion-wise about its richest spot, and the enrichment from layer to layer is not abrupt but gradual. Rolling draws this onion out into a nest of long concentric pods. If any excessive segregation remains after cropping, these axial drillings, even though they do not reach the very richest of the segregate, are almost sure to detect very marked segregation in a billet or bloom from at least one ingot of a heat; and this detection in any one piece should lead to the rejection of the whole heat, or at least to closer scrutiny.

If the maker knows that inspection is thus to be made with light and knowledge, he will see that the chance of detection of any harmful segregation is so great that his interests should compel him to crop liberally.

To be specific, I suggest a clause somewhat as follows:

§ 79. "*Check-drillings.* In addition to the regular drillings taken from the ladle-test for analysis, check-drillings shall be taken from the parts suspected of being most strongly segregated; for instance, from the axis of each end of the two blooms, billets or slabs which come from the upper end of the ingot remaining after cropping. Drillings from each such place shall be analyzed separately, and all the steel of the heat from which said drillings have come shall be rejected in case

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either the carbon or the phosphorus in said drillings exceeds by one-third the limits for those elements set for the ladle-test, or in case the sulphur in said drillings exceeds by two-thirds the limit for sulphur set for the ladle-test.

Instead of absolute rejection, it may suffice in some cases to provide for further and more rigid search for segregation. For instance, the upper two blooms, etc., may be rejected, and axial check-drillings may then be taken from the next two blooms, etc.

The number of blooms, etc., of which axial drillings are thus to be taken should, moreover, depend upon the relation between the size of these blooms and that of the ingot from which they are cut. If the ingots are very large and the blooms unusually small, then more blooms should be inspected in order to make sure that the inspection has really reached below the richest of the segregate. Indeed, what I have here advised is *pro forma*, to be adjusted to the needs of each case.

§ 80. *Further precautions in engineering specifications as to piping and segregation.* Although the precautions just described, inspection at the rolls and taking axial drillings, seem to me both searching and reasonable, effective for the buyer and acceptable to the honest and competent maker, yet both for those who do not agree with me and for those who wish additional precautions, we may next ask what further steps are reasonable.

They may be divided into two classes:

I. Cropping, in order to remove the piped and segregated parts, and

II. Restraining piping and segregation, and raising the pipe and the segregate towards the top of the ingot.

§ 81. *Cropping.* Though, as regards segregation, the axial drilling discussed in sections 78 and 79 is a searching test, yet neither it nor cropping until the cropped section looks solid gives really good evidence that the harmfully-piped parts have been rejected. That axial drilling does not is self-evident; and that continuing to crop until the cropped section looks solid does not is clear on the least reflection. The walls of the lower part of the pipe are usually pretty smooth, so that in rolling they fold down and close together. If they are unoxidized, they will weld; but if, as usually happens, enough atmospheric oxygen filters in through the top of the ingot to oxidize these walls, then they are likely not to weld. Yet they

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will be folded together so well that their unweldedness—*i.e.*, the fact that the pipe really constitutes a serious and persistent flaw—may not be apparent to the inspector, who is necessarily at some distance from the cropped section which he is inspecting, because it is so hot that he cannot hold his face near it.

Thus the apparent soundness of the hot cropped section gives no strong evidence that the cropping has gone deep enough to remove the harmfully-piped part, and of course it throws no trace of light on the question whether the cropping has gone deep enough to remove all harmfully-segregated parts, because the segregation usually lies far below the pipe.

Therefore, as a precaution against the presence of harmfully-piped parts, and as an additional precaution against that of harmfully-segregated parts, the buyer's engineer may reasonably specify, in important cases, that a certain predetermined percentage of the ingot's length shall be cut off, and that he shall inspect this off-cropping. This percentage should be based on the conditions of casting, and we should learn whether it is enough by examining certain individual ingots exhaustively, to see how deep in them harmful piping and segregation actually reach.[35]

§ 82. *Precautions in engineering specifications to restrain piping and segregation, and to raise the pipe and segregate towards the top of the ingot.* To recapitulate the steps already considered:

The pipe is shortened, though probably at the cost of increasing the degree of segregation:

(1) By casting in wide ingots;

(2) By casting in molds of low conducting power—*i.e.*, lined with sand or clay—especially if pre-heated.

The pipe is shortened and the segregate raised:

(3) By top-casting instead of bottom-casting;

(4) By slow casting;

(5) By casting with the large end up instead of down;

(6) By retarding the cooling of the top by means of a sinking-head or otherwise;

[35] In several of the cases reported the very richest of the segregate lies nearly 30 per cent. below the top of the ingot; indeed, in one extreme case it lies nearly 60 per cent. below the top. Again, Dr. P. H. Dudley points out that "In many cases the A rail of the ingot—the top—will be sound and the next, or B rail, include a local portion of unsound metal which visual inspection fails to detect." (*Proceedings of the New York Railroad Club*, Nov. 16, 1906, p. 527.)

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(7) By permitting deep-seated blow-holes to form through adjusting the quantity of silicon and manganese or their equivalent;

(8) By liquid compression.

The degree of segregation is lessened:

(9) By quieting the steel by adding aluminum or its equivalent;

(10) Probably by casting in small instead of in large ingots; and perhaps by steps which hasten the solidification, such as

(11) Casting at as low a temperature as practicable;

(12) Casting in thick-walled iron molds; and

(13) Casting slowly.

Of these, (2), (5), (6), (8), and (10) are not in such general use that it would be wise in most cases to call for them, because this would be likely to exclude many competent bidders, facilitate collusive bidding, and thus tend to raise the bids; and (1), (4), (7), (9), (11), and (13) can hardly be specified intelligently with the knowledge which the engineer has or can command at present; and even if they could, to specify them in effective detail would be to dictate to an unwelcome and unreasonable, indeed in certain cases to an intolerable degree, the details of the steelmaker's procedure.

But (3), top-casting, may reasonably be required in many cases, because it is not the exception but the rule, and because, in the very great majority of steel-works, to use it would involve no hardship, difficulty or considerable expense.

If we turn now from the common run of engineering specifications to those which not only are of unusual importance but also call for steel of unusual excellence, such as high-carbon steel for the wires of suspension bridges, we find three more of these precautions: casting (5) with the large end up, (6) with a sinking-head, and (10) in small ingots, not larger than 8 in. square, which may be considered very seriously. They are already familiar to most competent makers of such steel, and, indeed, are actually used by many of them. And, as the great value of these precautions in lessening the needed cropping and in raising the permissible limit of phosphorus and sulphur becomes better known, and as they thus become more

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generally adopted, the proportion of cases in which they may reasonably be called for will increase.

Of the other precautions, *permitting deep-seated blow-holes to form*, (7), is already in very general use. Though the buyer may hardly go so far as to demand that there shall be blow-holes, their presence in small quantities is to his advantage, and therefore should not only be permitted, but even welcomed, both in castings and in ingots, in the former provided that they are so placed that they do not weaken the castings materially, in the latter provided that they are so deep-seated that their sides do not become oxidized, and therefore are not in danger of imperfect welding.

Appendix.

Segregation and liquation. The established meaning of liquation is the extraction of a more-fusible from a less-fusible metal, by heating them to a temperature between their respective melting-points, so that the former melts and runs out from the latter. But this word has been so often used by careful metallurgical writers to mean also the local coalescing, discussed in § 56 B, p. 80, as distinguished from axial segregation, that it seems well to recognize this meaning, which I hereby call to the attention of the Institute. Indeed, this local coalescing is a first step in the process of bodily extracting the more-fusible metal from the less-fusible one. Before the more-fusible metal can run out from the other, it must first coalesce locally into particles of considerable size. The trend of custom seems to be strongly in the direction of using "segregation" in the sense of axial segregation and "liquation" in this special sense of local coalescing. Thus used these two words facilitate the discussion of this most important subject.

The objection to this step is that "liquation" would then have two meanings, differing in degree. But the old and established meaning of "bodily extraction" is to-day of little use. There is to-day little need of a word to designate the "bodily removal" process, but there is great need of a brief and clear word to designate local coalescing.

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The Influence of the Conditions of Casting on Piping and Segregation, as Shown by Means of Wax Ingots.

BY H. M. HOWE AND BRADLEY STOUGHTON, NEW YORK, N. Y.*

(New York Meeting, April, 1907.)

THIS paper presents an experimental verification of some of the predictions made by one of us[1] concerning the influence of certain conditions of casting upon the size and position of the pipe, and the position of the segregate, in steel ingots. The predictions which we have been able to verify are the following:

A. That the pipe is shortened and the segregate raised:

1. By slow casting;

2. By casting with the large end up instead of down;

3. By retarding the cooling of the top, *e.g.*, by means of a sinking-head.

B. That the pipe is shortened by slow cooling.

C. That the pipe and segregate lie in the last-freezing part.

Our procedure was to cast ingots of wax (commercially pure stearic acid) containing a little bright green copper oleate (usually 1.5 per cent.) under varying conditions; to saw each ingot open along a longitudinal plane passing through its axis; and to examine the longitudinal section thus laid bare. In order to make the segregate or enriched part more distinct in color from the impoverished part, a very little red cerasine was added. An experiment which will be described later showed that the cerasine itself does not segregate markedly, if at all. Hence we may assume that the strong contrasts between the green and red shown in so many of our ingots are due, not to segregation of cerasine, but to that of the green oleate, in the wax which in itself is uniformly reddened with cerasine.

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* Respectively Professor and Adjunct Professor of Metallurgy at Columbia University in the City of New York.

[1] Piping and Segregation in Steel Ingots, by Henry M. Howe, this volume, pp. 3 to 108.

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The wax ingots themselves, shown while presenting this paper, of course have not been tampered with in any way. But the contrasts between the red and green, so beautiful in these originals, make so little showing in a direct photograph that, in order to show them in our *Transactions*, we had to paint a photographic print so as to strengthen them, and then re-photograph this print. This, in turn, is because the difference in photographic value between the green and the red is so slight.

Taking up the evidence in detail, the influence of the rate of casting is shown in ingots Nos. 1, 2 and 3, Figs. 1, 2 and 3. Of these we cast Nos. 1 and 2 from the same casseroleful, half of which we transferred to a second casserole just before casting. We began the casting of these two at the same instant; but whereas the casting of No. 1 was finished in 30 sec., that of No. 2 was so slow that, though it was continuous except for momentary interruptions for heating the wax, it lasted 1 hr. and 13 minutes.

The pipe in the fast-poured No. 1 stretches down 90 per cent. of the ingot's length, and, except for some very thin bridges, is practically continuous for 49 per cent.; whereas in the slowly-cast ingot the pipe stretches down only 14 per cent. of the length of the ingot. In this particular ingot (No. 2) there is a second rudimentary pipe near the bottom, caused by our accidentally pouring at first faster than we intended. In ingot No. 3, which was poured slowly from the start, this second pipe is absent.

The segregate in the fast-poured ingot No. 1, though not strongly marked, can be traced at *A*, Fig. 1, as a dark pear-shaped mass, near the bottom of the ingot. The slowly-cast ingot No. 2 has rather a succession of local axial horizontal segregates than a single segregate, as is easily understood from the extremely slow rate of casting. In the still more slowly cast ingot, No. 3, these local segregates are so slight as almost to escape notice, and there is no noticeable single segregate.

The effect of casting with the large end up instead of down is shown in Figs. 4 and 5, which represent two ingots cast in immediate succession and under otherwise like conditions from the same casseroleful.

The pipe stretches down only 30 per cent. of the ingot's length when the large end is up, but 82 per cent. when the

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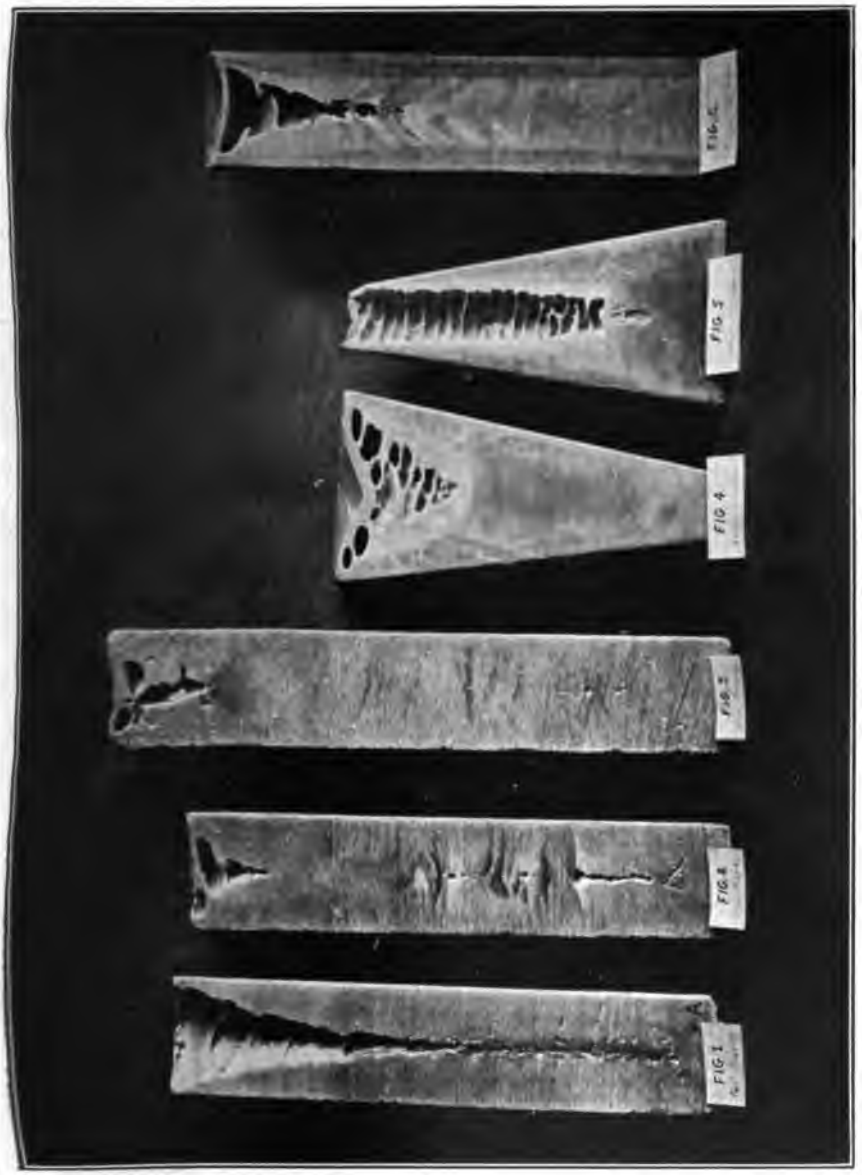

FIGS. 1 TO 6.—PIPING AND SEGREGATION AS SHOWN BY WAX INGOTS.

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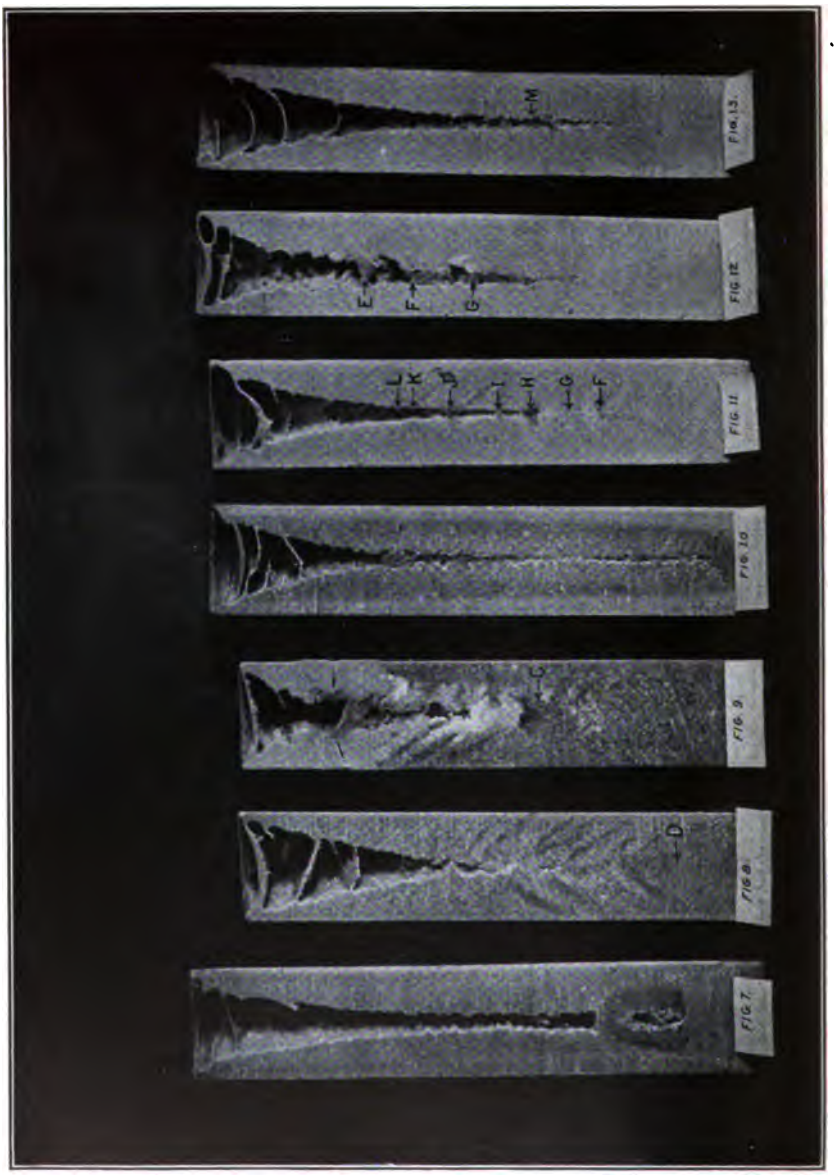

FIGS. 7 TO 13.—PIPING AND SEGREGATION AS SHOWN BY WAX INGOTS.

Ingots Nos. 14 and 15 have been omitted from the engraving.

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large end is down, and in the latter case is a large and practically continuous cavity for 69 per cent. of the ingot's length, save for the fairy-like transparent bridges.

The segregate lies well above the center in the ingot with the large end up, but very near the bottom in that with the large end down.

The difference in size between the top and bottom of these ingots, *i.e.*, their taper, is very much greater than in practice, with the purpose of exaggerating the effect of the direction of taper so as to make it clear to the eye.

The effect of retarding and of hastening the cooling of the top of the ingot is shown in Figs. 6 and 7, and in Figs. 8 and 9. The preparation of ingots Nos. 6 and 9 is described on p. 119.

The depth to which the pipe reaches as a nearly continuous cavity is only 26 per cent. of the ingot's length in the hot-topped ingot No. 6, but 85 per cent. in the cold-topped No. 7, and the extreme distance through which it can be traced is only 37 per cent. in the hot-topped against 85 per cent. in the cold-topped one. To put this in another way, if steel ingots should pipe like these, the part in which the pipe would be so oxidized that it certainly could not be welded would be only 26 per cent. in the hot-topped ingot against 85 per cent. in the other.

The segregate lies well below the middle of the pipeless part of the cold-topped ingot No. 8, but well above that of the hot-topped ingot No. 9. This pair of ingots does not show so well as Nos. 6 and 7 the influence of the distribution of temperature on the position of the pipe, because the temperature-lag of the slowly-cooling part was extremely slight, whereas in Nos. 6 and 7 it was very great. It is this temperature-lag, rather than the time-lag, that determines the depth of the pipe. The reason why retarding the cooling of the top shortens the pipe is that the upper part remains molten and thus ready to feed down into the pipe when, late in the piping-period, the inner walls are drawing outwards because of their adhesion to the virtually expanded outer walls. In order that this downward-feeding shall have a marked effect in shortening the pipe, the upper part must remain soft enough to sag thus up to the time when a large amount of the outward-drawing of the lower part shall have taken place, *i.e.*, until the freezing of the

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lower part shall have become very far advanced. But this implies that the temperature of the lower part must be much below that of the upper part, or in short that there must be a large temperature-lag.

The effect of the rate of cooling is shown in Figs. 8, 9 and 10. The pipe in the fast-cooled No. 10 runs down the axis for 91 per cent. of the ingot's length, and then branches at 45° to right and left towards the ingot's corners, following the path of the junction of the columnar crystals like that so often seen in steel ingots. In the slowly-cooled Nos. 8 and 9, the pipe stretches down 61 and 45 per cent. of the ingot's length, or in both cases very much less than in the fast-cooled ingot.

We had great difficulty in preventing this pipe-lengthening effect of rapid cooling from being masked by the pipe-shortening effect of the slower cooling of the top than of the bottom which naturally accompanies rapid freezing; and, of the two, the relative rate of cooling of the top and bottom is likely to outweigh the absolute rate of cooling, in its influence on the depth of the pipe. It was easy enough to cool the lower part of the ingot quickly, by setting the mold in iced water; but under these conditions the top lagged far behind the lower part, because from the top the heat necessarily escaped only very slowly into the air. We could not cover the ingot-top with our iced water, because that is so much heavier that it would run right down through the molten wax. Covering the ingot-top with a cold object helped us but little, because as soon as the mass began to settle an air-space formed above the wax itself, and through this air the heat passed only slowly. With ingot No. 10 we finally hit on the device of exposing it to the outer air on a very cold winter's day, with a good wind blowing. This seemed to give us about as near an approach to uniform thermal conditions at the top and bottom as could be hoped for, though the cooling of the top must still have lagged somewhat behind that of the bottom.

In slow cooling, though we saw no way of making top and bottom cool at exactly the same rate, yet we could make the difference of rate extremely small, and we could readily reverse its sign, by varying the strength of the small gas-flames played against the top and bottom of a vertical closed iron cylinder, Fig. 16, within which stood our mold, separated from the walls

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of the cylinder by a wide air-space. This difference of rate between top and bottom we thus reduced so far that it no longer sufficed to mask the simultaneous pipe-shortening effect of the slow cooling itself, as is shown by comparing the pipe-lengths of the ingots Nos. 8, 9 and 10.

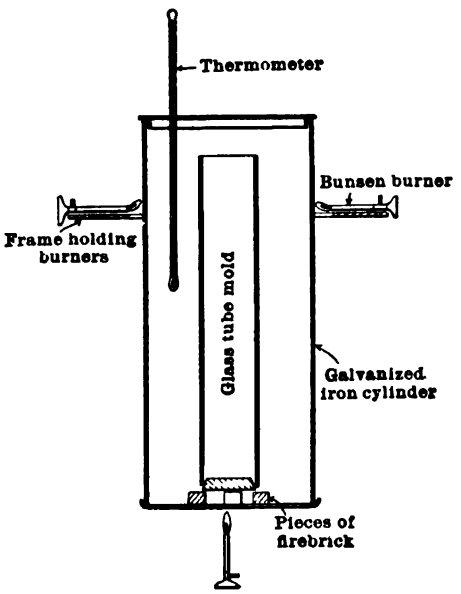

FIG. 16.—ARRANGEMENT FOR CONTROLLING THE RELATIVE RATE OF COOLING OF TOP AND BOTTOM, IN SLOW FREEZING.

Ingot No.	Rate of Cooling.		Length of Pipe, in Percentage of Length of Ingot.
	The Fastest Cooling was at	Rate of Ingots as a Whole.	
			Per Cent.
10	The bottom.	Fast.	96
8	The top.	Extremely slow.	61
9	The bottom.	Extremely slow.	45

Here the fact that the fastest cooling was at the top in No. 8 but at the bottom in No. 10 certainly tended to make the pipe deeper in No. 8 than in No. 10; nevertheless it was actually much shallower in No. 8 than in No. 10, so that the powerful effect of the change of position of the fastest-cooling point from top to bottom has here been greatly outweighed. What has outweighed it is clearly the difference in the absolute rate of cooling of

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these ingots, each taken as a whole. The explanation is that the slower cooling of No. 8 has shortened its pipe to a degree which greatly outweighs the pipe-lengthening effect of its cooling faster at top than at bottom. Here, then, we verify the prediction that slow cooling tends to shorten the pipe.

Segregation is much greater in slow than in fast cooling. It is true that there is a slight concentration of the oleate in the lower part even of the fast-cooled Nos. 10 and 11, and apparently in the irregular masses which cross the pipe of the former. But the difference in tint between the red impoverished parts and the green enriched ones is incomparably greater in the slowly-cooled Nos. 8 and 9 than in the fast-cooled Nos. 10 and 11, both transversely and lengthwise. Indeed the lengthwise difference, which is very slight in the fast-cooled ingots, is so great in the slowly-cooled ones that in the upper quarter of one of them, No. 9, the downward expulsion of the oleate looks complete to the eye.

That the last-freezing part contains the pipe and segregate is shown primarily in Fig. 12, and incidentally in Figs. 4 and 5, 11 and 13, and 8 and 9.

The ingot shown in Fig. 12 had its cooling hastened on its right-hand side by circulating cold water beside it, and retarded on its left-hand side by coating it with flannel, with the purpose of shifting the last-freezing part from the axis to the left. As we foresaw, this has shifted the pipe distinctly to the left of the axis.

The segregate, too, seems to have shifted so that it is concentric with the pipe. The bridges *E*, *F* and *G* are of so much darker and stronger a green than any of the rest of the ingot that we believe that they are the central axial part of the segregate; but the indications are not conclusive.

That the segregate lies in the last-freezing part is further shown by the facts (1) that it lies well above the center of the ingot No. 4 with the large end up, but far below the center in ingot No. 5 with its large end down; (2) that it lies far above the center of the pipeless part, at *C* in hot-topped ingot No. 9, but at *D* below the center of the pipeless part of cold-topped ingot No. 8; and (3) incidentally by a curious and interesting phenomenon shown in several of our ingots, but particularly well in Nos. 11 and 13. In the middle and lower part of

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No. 11 there are five bridges, of which the lowest, *F*, is rudimentary, and each successive one is better and better marked as we pass upwards, until at *I* we get a distinct bridge, and at *J* one which seems to be over-ripe in a special sense. We venture to call these "surface-tension bridges," a term which we will now explain.

The pipe, as one of us has shown,[2] is due to the more rapid cooling and contraction of the inner than of the outer walls of the ingot in the latter part of the freezing. The down-sagging of the upper part of the mass tends to make the pipe narrower and narrower as we pass down the length of the ingot: hence the narrowing of the pipe at its lower end, and the gradual cessation of the rudimentary pipelets, for instance in Fig. 11.

The bright green of the copper oleate in this ingot shows that there are many local centers of segregation. The length of our ingots is so much greater than their width that the whole of the segregate does not usually coalesce into a single mass, but instead there are several local centers of coalescence along the axis of the ingot. Now it seems to us that each of these local segregates causes a local surface-tension bridge. That is to say, the crack which, as the beginning of the pipe, starts near the ingot's top and travels downwards along the axis, is interrupted at a point where the local enrichment of oleate has made the wax so fusible, soft, and plastic, that it stretches out like so much molasses candy under the strain, and thus forms a surface-tension bridge, instead of cracking open like the rest of the wax above and below, which, because it has less oleate, is less fusible, and therefore has progressed farther in the passage from the molten to the rigid state.

In evidence of this we point to the fact that each of these bridges from the rudimentary *F* to the over-ripe *J* is not only greener but very much greener than the wax between the bridges. Moreover, at *K*, *L* and other spots we find not bridges but promontories of like verdancy, which look as if they had passed through the stages now represented by *H*, *I* and *J*— as if they had once been bridges, but had later been torn apart by the widening and down-sagging of the pipe, and that, now left protruding as green promontories, they are really relics of former bridges.

[2] This volume, p. 14, § 9 *et seq.*

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Like phenomena can be traced in ingots Nos. 13 and 15, though not so clearly. Had we No. 13 alone, all that we could assert would be that the very walls of the pipe in and above *M* were very much greener than any of the rest of the mass, and that the ruined bridges above also looked greener than the surrounding wax. But when we come to interpret the phenomena here and in many of the other ingots in the light of what we see so very clearly in ingot No. 11, there seems to be no reasonable doubt that these promontories and like masses are ruined surface-tension bridges.

Segregation or Stratification?

Without better knowledge of the relations of copper oleate and stearic acid, one might naturally ask whether we really have here a case of true segregation. May not the downward concentration of the copper oleate be only the subsidence of a mechanically suspended heavy foreign body, instead of the true segregation of a dissolved impurity? And may not the concentration of red at the top of our very slowly cooled ingots be due, not to the expulsion of the green copper oleate by segregation, as we have hitherto assumed, but to the upward concentration of the red cerasine, either by segregation or by stratification? We will now offer some evidence on these points.

To dispose first of the simple question whether the red cerasine concentrates upwards, we show ingot No. 14, made of stearic acid without copper oleate, but with the usual quantity of cerasine, and held molten for 1 hr. 50 min., to allow ample time for stratification if it does actually tend to occur, and then cooled quickly without disturbing the ingot or mold in the least, so that any stratification which should arise might persist. In spite of this ample opportunity for stratification none seems to have occurred, because the red appears to be uniform over the whole longitudinal section of the ingot. Hence we infer that the contrasts of red and green in our other ingots cannot be due to stratification of the cerasine.

Turning, now, to the question whether the concentration of the green oleate in the slowly-cooled ingots is due to stratification or to segregation, we have to admit at once that, in several of our ingots, a small quantity of copper oleate seems to have remained undissolved, and to have sunk to the bottom

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of the ingot. But this applies to only a very small quantity of the oleate, and this undissolved matter is in general readily distinguished from the segregate proper, because it forms sharp grains of considerable size right against the bottom of the ingot.

Now, the mere fact that the oleate in general lies well below the center of the ingot may be explained either by the simple stratification of a heavy suspended solid, or by the greater density of the segregate due to its enrichment in the heavy copper oleate. And even the fact that in certain cases the segregate lies above the center of the ingot (the hot-topped ingots and that cast with the large end up), can readily be explained on the stratification hypothesis by supposing that the concentration by gravity was more complete in the upper than in the lower part of the ingot, because the upper part remained molten longer, giving stratification a longer time to complete itself.

But this stratification-explanation breaks down when we consider the evidence in more detail. Thus the upper parts of ingots Nos. 6 and 9 remained molten and quiescent for a very long time (1 hr. 4 min., and 1 hr. 45 min.), so that, if the impoverishment of the upper part of No. 9 was due to mere subsidence of a suspended solid, the upper part of No. 6 should have been impoverished to a comparable degree. But the impoverishment in oleate, while in the upper quarter of No. 9 it has gone so far as to approach complete expulsion, in No. 6 is very slight. The difference in impoverishment in these two cases is far too great to be explained by the stratification-hypothesis, but it is just such a difference as should result from the great difference in the rate of freezing, which was extremely rapid in No. 6, but extremely slow in No. 9.

No. 6 was cast in an iron mold standing in a large empty tub, Fig. 17. Four very gentle flames from horizontal Bunsen burners were played against the lower part of the mold, and the frame which carried these Bunsen burners was raised slowly as the tub was gradually filled with iced water, the surface of which kept pace with the Bunsen burners slightly above it. During the very slow upward travel of the water and the flame, the upper part of the ingot had over an hour in which the copper oleate, had it been mechanically suspended, would have sunk downwards; but, when any given horizontal layer of wax once began to freeze, it was rapidly frozen by the iced water.

No. 9, on the other hand, was surrounded by a hot-air jacket, as in Fig. 16, slightly hotter above than below, and the temperature of this jacket as a whole was very gradually lowered, so that there was not only a slow approach to the

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freezing-range, but a very slow passage through that range, instead of the rapid passage which took place in No. 6.

As we pass inwards from the outer shell of ingot No. 6, we find first a dark band in which the freezing seems to have been so rapid as to restrain segregation very greatly. This is followed by a band in which the red predominates, and this in turn by the axial region in which the green predominates. Here we have exactly the sequence which we find in a steel ingot: an outer region but little impoverished; then a region of maximum impoverishment, followed by the axial region of maximum enrichment.

That even a true segregate should, if it is denser than the mother-mass, concentrate towards the lower part of the ingot, is only natural, as one of us has explained in case of the deep-seated segregate in ice ingots.[3]

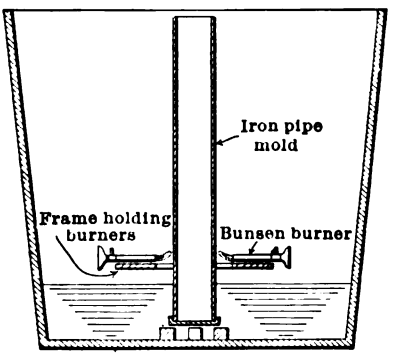

FIG. 17.—HOT-TOP CASTING ARRANGEMENT. THE FREEZING PROCEEDS FROM BELOW UPWARDS.

The green surface-tension bridges already noticed are further evidence in this same direction; and so are the diagonal streaks which we will consider later.

Though this reasoning seemed to us persuasive, we supplemented it by a direct experiment, the results of which seem absolutely irresistible. Stratification is a thing which occurs both before and during freezing, but segregation occurs only during freezing. Therefore, if we hold the mixture of wax, oleate, and cerasine molten for a very long time, longer than in any of the preceding cases, so as to afford ample time for stratification proper, and then cool it rapidly past the freezing-point so as to leave but little time for segregation; then if our phenomena are really of stratification, the upward concentration of the red and the downward concentration of the green

[3] This volume, p. 68.

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should be extremely strong in the resultant ingot. If, on the other hand, our phenomena are not of stratification but of segregation, then there should be only the moderate color-contrasts which the other fast-cooled ingots show.

Therefore we held ingot No. 15 molten for 1 hr. 50 min., or longer than in any of the other cases, in an iron mold within a closed air-chamber, as in Fig. 16; and then, without moving the mold itself or otherwise disturbing any stratification which might have occurred, we froze the wax rapidly by pouring cold water into the air-space which surrounded the mold. The water rose smoothly in this outer air-space, and there was nothing to disturb the wax in its mold. As may be seen, there is only the slight color-contrast which the other quickly-cooled ingots show, and nothing resembling the extreme downward concentration of the green and upward concentration of the red which the slowly-frozen ingots, such as Nos. 8 and 9, show so strikingly.

The inference seems to us irresistible that the color-contrasts in these slowly-frozen ingots are due not to stratification but to true segregation. Hence it further seems that the evidence which this paper offers as to the influence of the several variables on segregation is true and valid evidence.

The Diagonal Streaks.

Early in this investigation we noticed curious diagonal streaks stretching inwards and upwards from the outside of certain ingots. Nos. 4, 6, 8 and 11 show these streaks in what we think nearly their natural position, and Nos. 5 and 9 show them distorted by sagging, as we will now try to explain. For brevity we will describe these in positive terms, but with the understanding that we put forth our explanation tentatively in spirit.

The red diagonal bands are the first-freezing crystals, which stretch out into the molten mother-wax. In freezing, they naturally reject part of their dissolved copper oleate, which thus concentrates in the spaces between them. They probably at first shoot out at right angles to the sides of the ingot, the cooling-surface; but from this horizontal position their inner ends swing upwards, because they are lighter than the molten wax into which they protrude, certainly because they contain less of the heavy copper oleate, and perhaps also because of the expansion of the wax in freezing. That part of the heavier

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TABLE I.—*Details of Preparation and Description of the Wax Ingots.*

Ingot No.	Laboratory No.	Preparation.						Description.			
		Quantity of Copper Oleate.	Length.	Diameter.	Material of Mold.	Mold Surrounded By:	Rate of Cooling.	Rate of Pouring.	Depth of Pipe in Percentage of Length of Ingot.		Depth of Center of Main Segregate in Percentage of Length of the Ingot.
									Total Depth.	Depth of Continuous Part.	
		Per ct.	In.	In.							
1	21	1.5	9½	2	Iron.	Cold water.	Fast.	30 sec.	90 ±	49 ±	Uncertain.
2	22	1.5	9¼	2	Iron.	Cold water.	Fast.	1 hr. 13 min.	14	14	66 ±
3	34	0.75	10¼	2	Iron.	Cold water.	Fast.	1 hr. 39 min.	18	13	56
4	24	1.5	6¾	*a* Large end up. *a* Large end down.	Wood.	Air.	Intermediate	50 sec.	30	30	41
5	23	1.5	6¾		Wood.	Air.	Intermediate	55 sec.	82	69	94
6	37	1.5	9	2	Iron.	Ice-water.	*b* Fast. Top fast, bottom slow.	30 sec.	37	26	94
7	26	1.5	10⅛	2	Glass.	*c.*	Very slow, 1 hr. 40 min.	40 sec.	85	85 *d*	85 ±
8	38	1.5	9⅝	2	Glass.	Hot-air jacket.	Very slow, 1 hr. 45 min.	48 sec.	61	47	86 ±
9	39	1.25	9⅝	2	Glass.	Hot-air jacket.	Very fast, about ¾ hr.	35 sec.	45	21	59 ±
10	35	1.5	10⅛	2	Iron.	Air with wind.	Very fast.	25 sec.	96	33 ±	34 ±
11	15	1.5	10¼	2	Iron.	Ice-water.	Very fast.	2 min 55 sec	80	45	84 ±
12	27	1.5	10⅛	2	Glass.	*c.*	Fast on one side, slow on other side.	40 sec.	72	30	31 ±
13	18	2.5	10½	2	Iron.	Ice-water.	Very fast.	51 sec.	81	56	62 ±
14	40	0	9¾	2	Iron.	Running cold water.	Fast, 1 hr.	50 sec.	53	53	.....
15	41	1.5	9¾	2	Iron.	Running cold water.	Fast, 1½ hrs.	55 sec.	85	37 ±	38

a Pyramidal ingots, 3¼ in. square at the large end and 1 in. sq. at the small end.

b Hot top. The mold, which stood within a tank, was heated near its lower end by four small gas-jets, which were very slowly raised, while at the same time iced water was run into the tank just fast enough to keep its upper surface at a slight but nearly constant distance below the gas-jets. Thus the freezing at any given level was very rapid, but the freezing-zone traveled upwards very slowly. From the time the iced water rose to the bottom of the mold, till the time when it reached the top of the mold, was about 1 hr. Thus we forced the freezing to take place from below upwards.

c Cold top. The upper part of the mold was cooled by jacketing it with a movable coil of rubber pipe, through which cold water circulated, while the lower part was warmed by jacketing it with a fixed coil of wire heated by electric resistance. The cold jacket was lowered gradually from the top of the mold to within about 2 in. of the heating-jacket, a travel of about 6 in., in 1 hr. 15 min. Thus we forced the freezing to take place from above downwards.

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molten mother-wax which is entangled or engulfed between these upward-pointing early-freezing growths, may thereby be prevented from running down to join the rest of the segregate in the lower part of the ingot; and, where opposite branches meet in the axis of the ingot, the locally-imprisoned segregate later forms one of the surface-tension bridges.

But, if there is much sagging of the mass, these upward-pointing trees are more or less bent downwards by it, and this downward bending increases with the amount of sagging. Thus there could be but little sagging in No. 4, cast with the large end up, because the freezing proceeds from below upwards; and in No. 5, with its large end down, there is very great sagging, because freezing proceeds from above downwards, as the shallow pipe in No. 4 and the deep one in No. 5 prove. It is in perfect accord with our ideas that the streaks in No. 4 point more nearly vertically upwards than in any other ingot, and that in No. 5 they are much more nearly level. In the suddenly-cooled No. 11 these streaks point strongly upwards, a natural consequence of the rapidity of cooling, which has given little time for sagging to take place; witness the great depth of the pipe. In No. 8, though the slow cooling has given plenty of time for sagging, there has actually been but little because of the cold top, which has led to a deep pipe; and it is quite in accord with this that the streaks point strongly upwards in spite of the slow cooling.

Finally, in the slowly-cooled hot-topped No. 9, both the slow cooling and the fact that the upper part remained molten and free to feed down into the pipe, led to so much sagging as both to cause a short pipe and to overturn these initially upward-pointing trees, and make them point downwards. In this case the sagging was so extreme that the ingot shortened itself by about 5 per cent. Thus in every respect the position of these streaks agrees with our explanation of their nature.

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d The continuous part of the pipe reaches nearly to its bottom, and extends through the thick bridge which, in the lower part of Fig. 7, masks it.

e *Lateral deflection.* The cooling of the right-hand side of the mold was hastened by jacketing it with vertical rows of rubber tubing, through which cold water circulated for 1 hr. 10 min.; and the cooling of the left side was retarded by jacketing it with flannel. Thus we shifted the last-freezing region from the axis towards the left.

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We have to thank Dr. Parker C. McIlhiney most warmly for aid in choosing these substances, copper oleate and cerasine, for showing these effects of segregation; and Mr. E. C. Groesbeck, S.B., for his careful and efficient execution of most of the manipulation.

An Early Instance of Blowing-In Without "Scaffolding-Down."

BY FRANK FIRMSTONE, EASTON, PA.

(New York Meeting, April, 1907.)

In the early decades of the past century the method of starting iron blast-furnaces by "scaffolding-down" seems to have been in universal use for coke-furnaces and, at least in this country, for charcoal-furnaces also. It was likewise used at some of the early anthracite-furnaces, as is stated in the important "Reminiscences" printed by the late Samuel Thomas.[1]

It had been generally superseded in Great Britain as early as 1850, by some variations of the plan now used, and was rarely used when Percy wrote.[2]

On the Continent it was still followed in the 60's,[3] and was elaborately described by Valerius in 1852.

I have never read any statement as to when and by whom "scaffolding" was first dispensed with, but the following extracts from journals of my father seem to fix an authentic date for an instance in this country:

"[Pittsburg] July 12, 1838.—Went with Major Wade to his foundry, etc., and he gave me a letter to D. Tyler, Esq., Farrandsville, Agent Boston Coke & Iron Co."

"[Lockhaven, Pa.] July 18, 1838.—Heard there was a flat-boat going to take some gentlemen up to Farrandsville. Got on boat and to Farrandsville, 5 or 6 miles, in 1 hr. 30 min. Dined, and then delivered my letter to Mr. Tyler, who showed me much attention, and everything belonging to the furnace, and had the water turned on the wheel that I might see the blowing-cylinders at work. There are two cylinders, 6-ft. stroke, 56 in. in diameter, but they find that one makes more than enough blast for the furnace. The furnace has 16-ft. boshes, 7-ft. tunnel-head, and is 47 ft. high, 3 tuyeres, and 3 heating-ovens [hot blast-ovens]. A

[1] *Trans.*, xxix., 914, 915 (1899).

[2] *Iron and Steel*, p. 491 (1864).

[3] De Vathaire, *Etudes sur les Hautes Fourneaux*, p. 143.

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most beautiful stack, the finest I ever saw. They have tried three times to start the furnace . . . and have failed each time; longest time of blowing, a week. They have now a person by the name of Perry from Staffordshire, who is going to start her."

On Oct. 28, 1838, being at Karthaus furnace, Clearfield county, which then was owned by Burd Patterson, Henry C. Carey and others, he writes:

"Professor Johnson, Philadelphia [Walter R. Johnson], arrived with a letter from Mr. Carey, in which he says: 'Professor Johnson visits Karthaus with a view to see what we have done and what we may do, and request you will give him every information, etc.' . . . Mr. Johnson says they blew in at Farrandsville and made 10 or 15 tons good grey-iron and blew out for want of water; that they filled her above the boshes with charcoal, then put on coke for a few feet and burdened her, and then blew off without scaffolding. He says their hearth is of fire-brick in segments, thinks about 18 in. by 12 in., and that the walls are 4 ft. thick."

This seems to establish beyond doubt that Perry, who afterwards blew in the anthracite-furnace at Pottsville,[4] used the present method of dispensing with "scaffolding" in the late summer or early fall of 1838. It is worth noticing that at that time the hearths were almost always built of sandstone, both here and in Great Britain.[5] One reason for the use of the tedious method by scaffolding was that the hearthstones were thereby more gradually heated up than by blowing at once, as we now do, and consequently were less likely to suffer by spalling. This reason failed, of course, when the hearth was built of fire-brick, as was the case at Farrandsville.

Whether Perry was the first to dispense with scaffolding, and whether this was the first occasion when the method was used, does not, of course, appear, but certainly it was practically unknown at that date.

I am inclined to think that Farrandsville was the first furnace in this country to make iron using both coke and hot blast.[6]

[4] *Trans.*, iii., 153 (1874–5).
[5] Percy, *Iron and Steel*, p. 477 (1864).
[6] Swank, *Iron in All Ages*, 2d ed., p. 369 (1892).

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Laboratory Experiments in Lime-Roasting a Galena-Concentrate with Reference to the Savelsberg Process.

BY H. O. HOFMAN, R. P. REYNOLDS AND A. E. WELLS, MASSACHUSETTS INSTITUTE OF TECHNOLOGY, BOSTON, MASS.

(New York Meeting, April, 1907.)*

I. INTRODUCTION.

LIME-ROASTING is a term proposed by Ingalls[1] for the operation of forcing air under pressure through a mixture of galena and lime at the kindling-temperature with the object of oxidizing lead and sulphur and of fritting or fusing the charge. If finely-divided galena were treated in this manner without the addition of lime, the heat set free by the oxidation of part of the lead and the sulphur would be sufficiently great to fuse undecomposed sulphide, and thus stop desulphurization. Besides the chemical action that the addition of lime, limestone or gypsum to the charge may have, the admixture has the physical effect that it keeps the particles of galena separated from one another and accessible to the oxidizing effect of the air.

At present, three methods of lime-roasting are carried out on a working-scale,—the Huntington-Heberlein, the Carmichael-Bradford and the Savelsberg[2] processes. In the last, which interests us here, an 8-ton charge is made up of galena, limestone and perhaps some siliceous or ferruginous flux; the whole is crushed to pass a screen with 3-mm. holes and moistened with 5 per cent. of water. It is fed gradually into a bowl-shaped converter, 6.56 ft. in diameter, supported by trunnions attached to a truck. On the bottom the converter has a grate with blast-inlet beneath. In starting, the truck with the converter

* Read by title at the London Meeting, July, 1906. Manuscript received Aug. 23, 1906.

[1] *Engineering and Mining Journal*, vol. lxxx., p. 402 (1905).

[2] United States Patent, No. 755,598, March 22, 1904. *Engineering and Mining Journal*, vol. lxxx., p. 1067 (1905) (Ingalls); vol. lxxxi., p. 9 (1906). *Mining Magazine*, vol. xii., p. 391 (1905) (Savelsberg). *The Mineral Industry*, vol. xiv., p. 407 (1905) (Hofman).

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is run underneath a stationary hood, which carries off the gases and fumes; the grate is covered with a layer of crushed limestone for the sake of protection; then follows a bed of glowing coal or coke, to be covered by a second layer of limestone to prevent contact of fuel and charge. A gentle blast is turned on and charge fed in to the depth of 12 in. Oxidation begins at the bottom and sulphurous gases are given off; when the roasting approaches the surface and this becomes red-hot, a second layer of charge is fed in, and feeding continued at intervals until the converter has been filled. While charging, about 247 cu. ft. of air is forced in with a pressure ranging at the start from 2.75 to 4.5 oz. per sq. in.; the volume of air is increased with the amount of charge fed, and this causes the pressure to rise to from 11.5 to 13.5 oz. per sq. in. toward the end of the blow, lasting about 18 hours. Desulphurization is followed closely by scorification and this by solidification. The charge does not become liquid as a whole, as the formation of the slag is a heat-absorbing reaction and as the blast chills the slag. The converter, when blown, is withdrawn on its truck from beneath the hood; the charge is dumped on to an upright iron bar to break it into several pieces, which are then further reduced in size by wedging and sledging. A typical charge at Ramsbeck, Westphalia, with 100 parts of galena-concentrate (Pb, from 60 to 78; S, 15 per cent.), 10 siliceous silver-ore, 10 spathic iron-ore and 19 limestone, averaging SiO_2, 11 per cent., will retain from 2 to 3 per cent. of S when successfully blown.

The literature of the process gives very little information upon the effects which variations in the addition of limestone and changes in the volume of blast may exert upon desulphurization, fusibility and losses in lead and silver. The aim of the experiments was to supply this lack as far as laboratory work could do it.

II. Experimental Work.

The materials used in the experiments were ore, limestone and quartz. The ore was a galena-concentrate from the Cœur d'Alêne district, Idaho. A screen-analysis gave the data presented in Table I., which shows that, with the exception of two grades, the 9- on 12-mesh and the through 100-mesh, the different sizes were evenly distributed. The ore was crushed to pass a 2.8-mm. screen.

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TABLE I.—*Screen-Analysis of Galena from Cœur d'Alêne District, Idaho.*

Size.	Weight.	Weight.
Mesh.	Kilograms.	Per Cent.
Through 6 on 9	6	26.1
Through 9 on 12	4	17.4
Through 12 on 16	1.3	5.7
Through 16 on 20	1.55	6.7
Through 20 on 30	1.9	8.3
Through 30 on 40	1.45	6.3
Through 40 on 60	1.4	6.1
Through 60 on 100	2.0	8.7
Through 100	3.45	15.0
	23.05	100.3

The chemical analysis of an average sample gave: S, 10.58; FeO, 5.95; Al_2O_3, 4.80; SiO_2, 18.58; Zn, 2.57; Pb, 54.70; CaO, 2.20; MgO, 0.40; Ag, 0.135 (= 39.50 oz. per ton); total, 99.915 per cent. Calculation sets forth that lead and zinc are covered by sulphur, being present as galena and blende, and that, in the absence of carbon dioxide, the gangue is made up of silicate or quartz and silicate. The limestone was a pure marble, which, upon analysis, gave: SiO_2, 0.26; CaO, 55.01; MgO, 0.85; CO_2, 44 (calculated); total, 100.12 per cent. The silica added to the charges to obtain various silicates was pure quartz, assumed to contain 100 per cent. of SiO_2.

Two converters were employed for blowing the charges, one a clay crucible, the other a slag-pot. The crucible-converter, in which most of the tests were made, is represented in Fig. 1: *A* is a Morgan clay crucible, size K, with a hole 0.5 in. in diameter drilled in the bottom to admit the air-blast; *B*, an iron funnel ending in a piece of gas-pipe, connected with a T-joint, *C*, one arm of which is closed by a screw-plug, *D*. The slag-pot converter, represented in Fig. 2, is a small-size detached Devereux slag-pot. The grate, *A*, is a cast-iron plate perforated with holes 0.25 in. in diameter; the blast-inlet pipe, *B*, is screwed into the tap-hole; *C* is the hood for carrying off gases and fumes; *D*, the charging opening, closed by a hinged door. The blast for the converters was furnished by a Root blower, No. ½; the pressure was measured by a water-gauge calibrated to read in inches. The temperature measurements were made with a Le Chatelier thermo-electric pyrometer.

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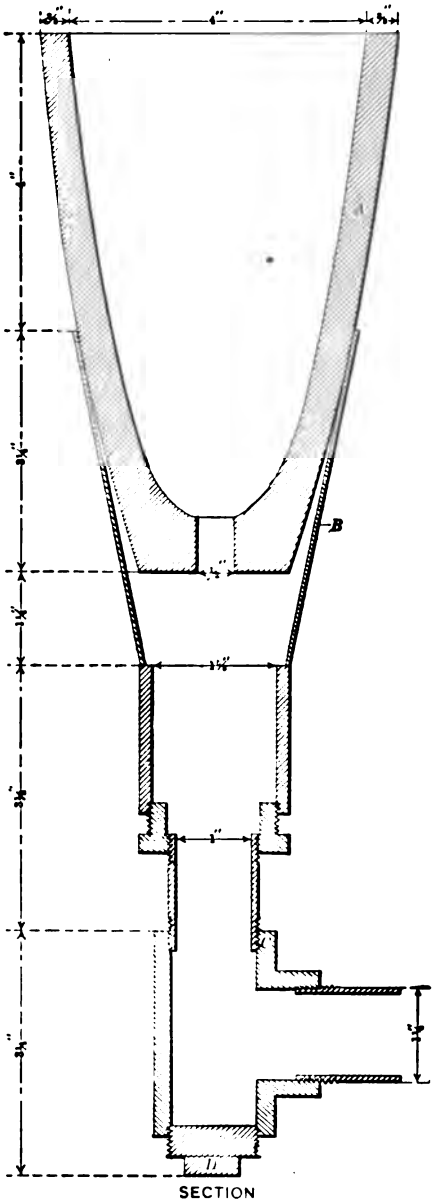

FIG. 1.—CRUCIBLE-CONVERTER.

Two series of tests were made in the crucible-converter, using 1 kg. of ore in each experiment. In the first series a singulo-silicate-slag was made the basis of calculation. Since the ore

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did not contain enough silica to form a singulo-silicate after the quantity of limestone suited for lime-roasting had been added, the deficiency was made up with quartzite. In the second series, the quantities of limestone used were similar to those in the first series, but no additions of quartzite were made; the

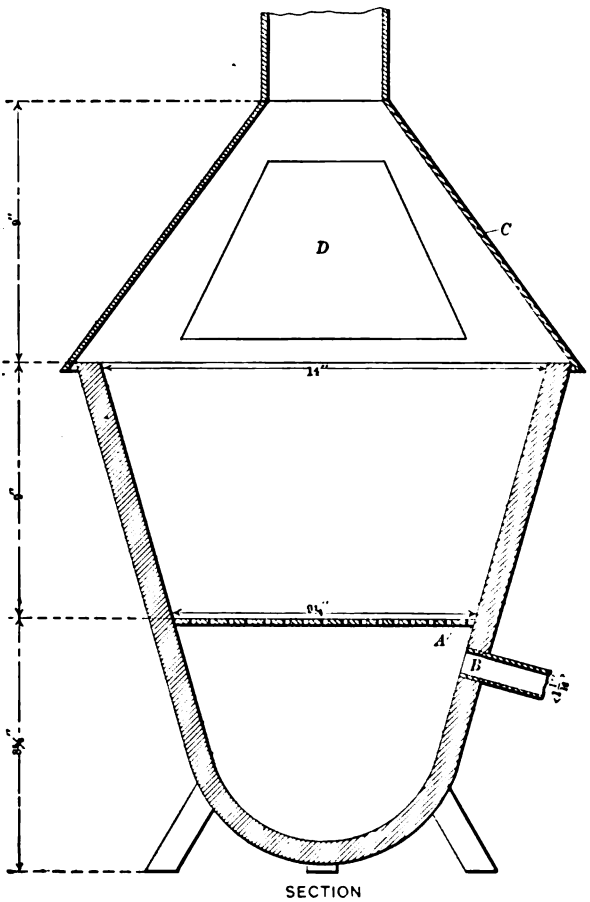

FIG. 2.—SLAG-POT CONVERTER.

products obtained were therefore subsilicates of varying silicate degree. In conclusion, two tests were made in the slag-pot converter with 20 kg. of ore to see whether the size of charge had any effect upon the result. The charges were fluxed according to the most favorable results obtained in the experiments with 1 kg. of ore.

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The modes of procedure in the small- and medium-size tests were similar. With the crucible-converter, the crucible was warmed in the ash-pit of an assay-furnace, the inside of the receiving-funnel was coated with a clay wash in order to make an air-tight joint, and the warmed crucible pressed into the funnel. A few pieces of limestone, nut-size, were placed in the bottom of the crucible to act as a grate; a thin layer of ignited charcoal was spread over the limestone, and the blast started. When the charcoal was burning freely, it was covered with a second layer of limestone, of a pea-size. The charge, mixed with enough water to give it the consistency of brasque, was then fed in gradually. A thin layer was spread over the limestone, and as soon as it had become ignited and glowing spots began to appear on the surface, a second layer was added, and so on until the crucible was filled. The blast was continued until no more fumes passed off from the surface. Any blow-holes that formed, here and there, were closed by adding fresh charge or by poking down the crater-like walls of the cavities. When a charge had cooled down somewhat, it was dumped out; the coarse limestone was picked off from the bottom, the slagged and unslagged portions were separated and weighed, the slagged part was crushed, added to the pulverulent part, and the whole sampled down and assayed for sulphur, lead, and silver. A crucible could be used for several charges, as these did not attack it; it had to be replaced, however, after a few runs, as it became fissured by a longitudinal or diagonal crack, caused apparently by the unequal expansion of the clay and the iron funnel.

The slag-pot converter was warmed by burning a thin layer of charcoal on the grate with a gentle blast; otherwise, the mode of procedure was the same as just outlined.

Temperature-measurements, by means of a Le Chatelier thermo-electric pyrometer, were made with five charges to see in what manner the rise in temperature took place, and to find the highest degree of heat attained in the process. Three continuous readings were taken with the crucible, and two with the slag-pot converter. The results are plotted in Figs. 3 to 7. With curves Nos. 3 and 4 the thermo-junction was fixed at about one-third the height of the crucible, with curve No. 5 at about two-thirds; with curves Nos. 6 and 7, representing the me-

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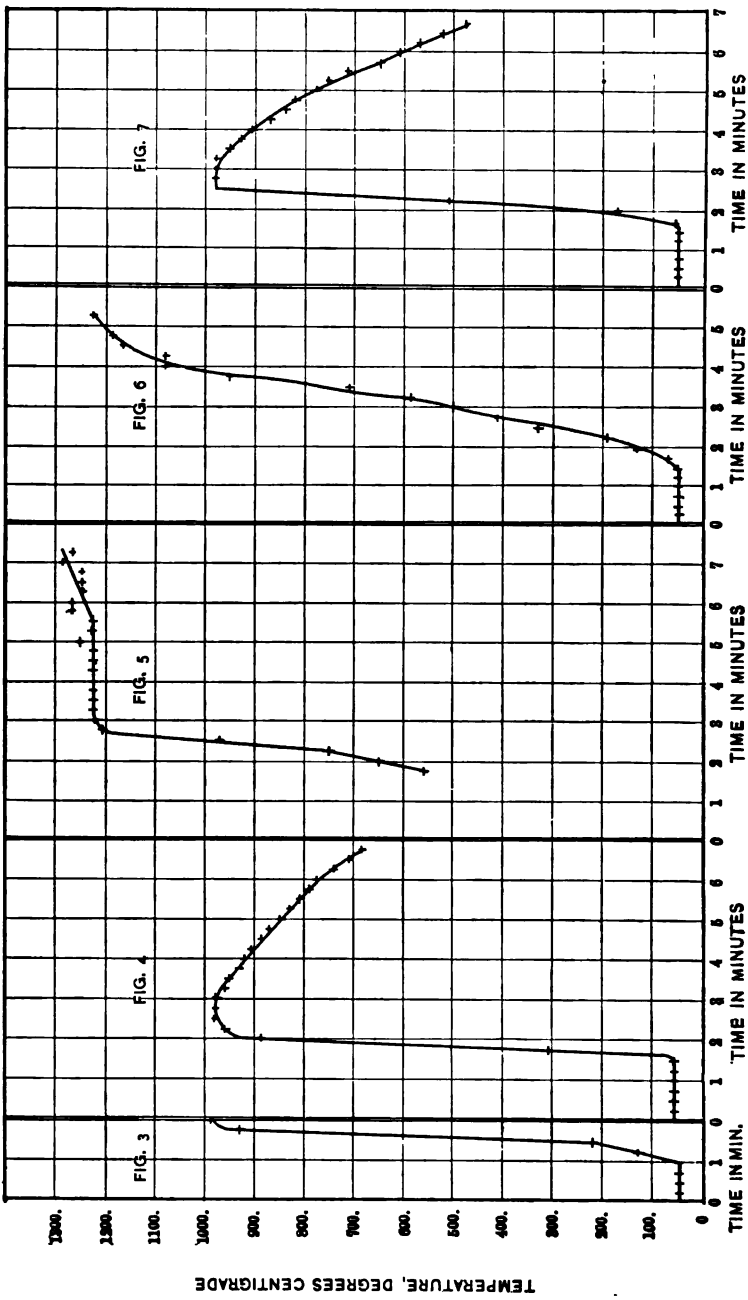

FIGS. 3 to 7.—TEMPERATURE-MEASUREMENTS IN CONVERTER.

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dium-size charges, the thermo-couple was held near the center of the slag-pot. In all cases temperature-readings were begun when the ore-charge reached the junction, and then continued at intervals of 15 sec. In curve No. 3 the two thermo-wires parted at 990° C., in curve No. 6 the galvanometer ceased to act at 1,220° C. for some unexplained reason, and in curve No. 5 there was a halt at the same temperature, followed by a rise after a few minutes' interval. These imperfect records are given, as they show, as far as they go, features similar to those of the complete records. The five curves all show a sudden quick rise of temperature to a maximum of 980° C. with the small charges, and to an apparent maximum of 1,225° C. with the medium-size charges. The subsequent gradual fall of temperature is seen only in curves Nos. 4 and 7.

III. Record of Results.

The results obtained from the 17 tests made are recorded in Tables II., III. and IV., and the data of the 15 small-size charges relating to the elimination of sulphur and the loss in lead and silver are represented graphically in Figs. 8 to 19.

Table II. and Figs. 8 to 11 represent the singulo-silicate series of tests. The weight of a charge had to be adapted to the size of the crucible; with small quantities of flux 1,000 g. was taken; with increasing quantities, the weight had to be reduced. The percentage of limestone of the charge ranged from 4.7 to 30, and the necessary silica varied accordingly. The mixtures were moistened in each case with 5 per cent. of water. The time it took to fill a crucible varied from 4 to 9 min., the blast being kept constant at 2 in. pressure. In two cases this pressure was maintained throughout, in four it was raised to 6 in., and in four to 10 in.

The runs lasted from 14 to 18 min. The blown product weighed less than the original charge, as was to be expected. The data giving the percentages of slagged and pulverulent parts of the blown charges show that limestone could form as much as 24 per cent. of the weight of the charge, and this when blown still furnish more than 95 per cent. of slagged material. When, however, this quantity of limestone was exceeded, and the lead in the charge reduced to less than 33.7 per cent., and the sulphur to less than 6.54 per cent., the heat generated was

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TABLE II.—*Singulo-Silicate Series.*

	Number of charge	1	2	3	4	4a	4b	5	6	6a	7
Charge.	Ore, grams	1,000	1,000	1,000	1,000	1,000	1,000	813	800	800	667
	Limestone, grams	50	120	150	200	200	200	217	276	276	325
	Silica, grams	...	17.8	33.3	52	52	52	55.4	72	72	88
	Water, per cent	5	5	5	5	5	5	5	5	5	5
	Limestone, per cent. in charge	4.7	10.6	12.7	16	16	16	20	24	24	30
	Time taken feeding, min	4	5.5	5	8	6	5.5	7	9	8	9
	Duration of run, min	8	16	14	17	15	12	16	18	17	12
Blast.	Pressure while charging, in. water	2	2.5	2.25	2	2	2	2	2	2	2
	Pressure, maximum, in. water	2	10	10	10	2	10	6	6.5	6	6.5
Product.	Weight, grams	930	1,015	1,055	1,105	1,117	1,115	970	1,035	1,038	945
	Slagged part, per cent	100	96.35	98.56	100	96.3	97.8	97.5	95.2	96.9	90
	Pulverulent part, per cent	00	3.65	1.44	00	3.7	2.2	2.5	4.8	3.1	100
Sulphur.	Sulphur in ore-charge, grams	105.8	105.8	105.8	105.8	105.8	105.8	86.0	84.6	84.6	70.56
	Sulphur in ore-charge, per cent	10.08	9.29	9.34	8.45	8.45	8.45	7.74	7.92	7.37	6.54
	Sulphur in product, per cent	6.73	5.22	4.3	3.75	3.77	3.81	3.56	3.61	3.46	3.58
	Sulphur in product, grams	62.6	52.98	45.36	41.44	42.1	42.4	34.5	37.4	35.9	33.8
	Sulphur eliminated, per cent	40.8	49.9	57.1	60.8	60.2	59.9	59.9	55.8	57.6	52.1
Lead.	Lead in ore-charge, grams	547	547	547	547	547	547	444	437	437	364
	Lead in ore-charge, per cent	52.1	48.1	46.2	43.7	43.7	43.7	40.9	38.1	38.1	33.7
	Lead in product, per cent	57.31	52.60	50.60	47.23	48.65	50.4	45.77	40.65	44.5	38.8
	Lead in product, grams	533	533.5	534	522	543	562	444	421	462	367
	Lead lost, per cent	2.55	2.37	2.37	4.55	0.73	gain	0.0	3.66	gain	gain
Silver.	Silver in ore-charge, grams	1.35	1.35	1.35	1.35	1.35	1.35	1.097	1.08	1.08	0.90
	Silver in product, grams	1.32	1.33	1.30	1.26	1.27	1.27	1.02	1.02	1.01	0.81
	Silver lost, per cent	2.2	1.48	3.78	6.67	6.67	6.67	6.4	5.55	6.5	4.4

insufficient to fuse the charge; this remained pulverulent, with here and there an agglomerated particle distributed through it. Slagged ore had the general characteristics of ore that has been agglomerated on the hearth of a hand-reverberatory roasting-furnace; the color, however, was more greenish-brown than black; it was tough, and the hardness increased as the additions of limestone decreased; wherever the elimination of sulphur was imperfect, streaks or bunches of undecomposed galena could be easily traced.

Curves Nos. 8 and 9, representing the percentage of sulphur remaining in the blown charge and the percentage of sulphur eliminated, show that in a charge the percentage of sulphur decreases as that of limestone increases until the latter reaches 20 per cent., and that from this point on the residual sulphur remains constant at 3.5 per cent. On the basis of percentage of sulphur eliminated, the maximum of 60 per cent. is reached with a charge containing between 16 and 20 per cent. of limestone, and then begins to decrease. In curve No. 10 the loss of lead increases from 2.55 per cent. with increasing limestone until 10.6 per cent. has been reached; it then remains constant to 12.7 per cent., when it drops quickly, reaching zero

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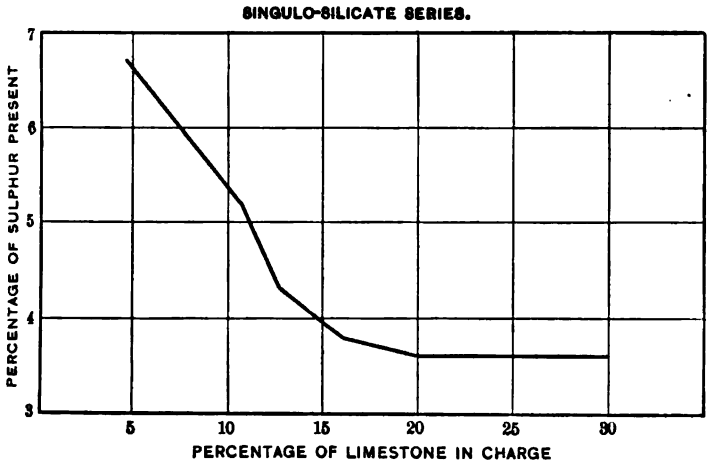

FIG. 8.—Residual Sulphur.

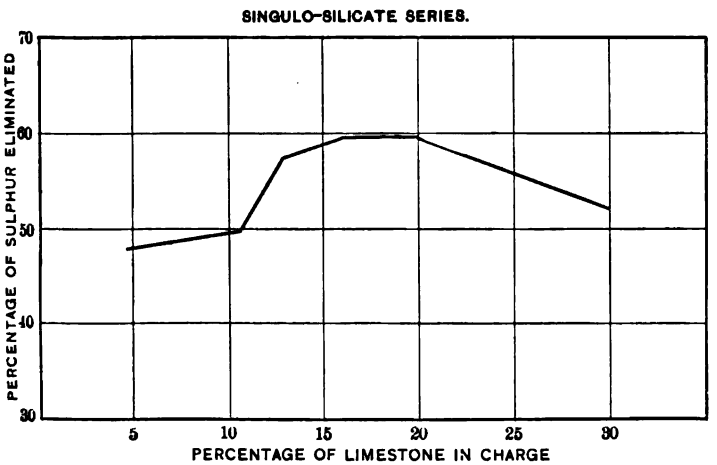

FIG. 9.—Eliminated Sulphur.

FIGS. 8 and 9.—RESIDUAL AND ELIMINATED SULPHUR.

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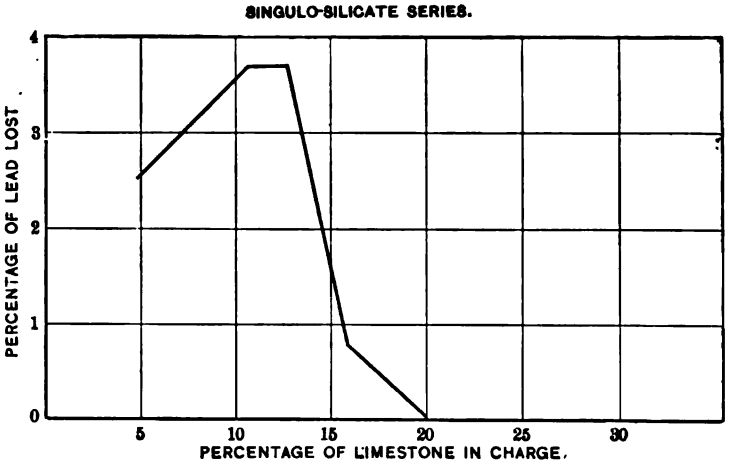

FIG. 10.—Loss of Lead.

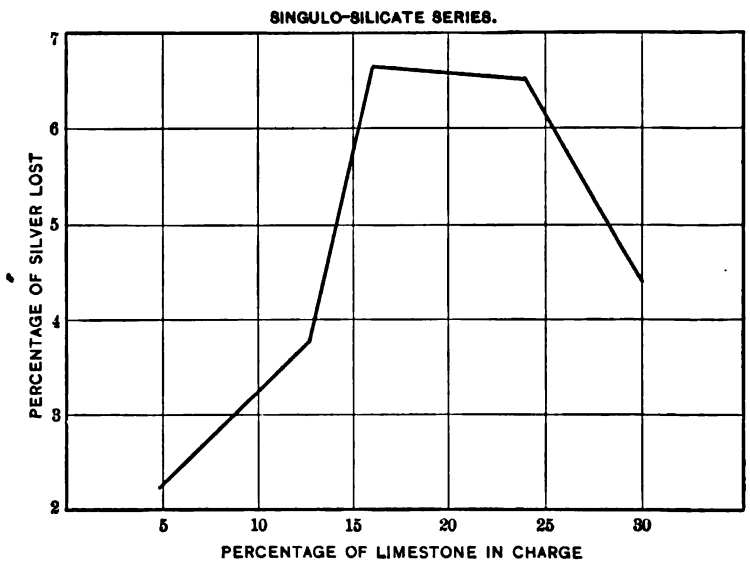

FIG. 11.—Loss of Silver.

FIGS. 10 and 11.—LOSS OF LEAD AND SILVER.

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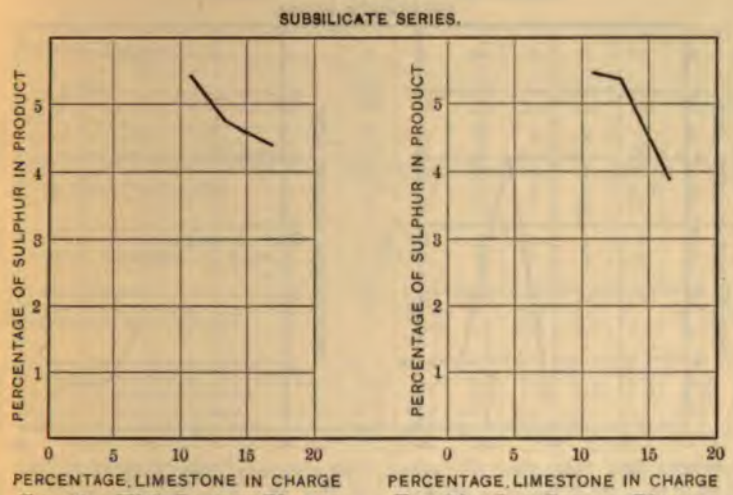

Fig. 12.—High-Pressure Blast.

Fig. 14.—Low-Pressure Blast.

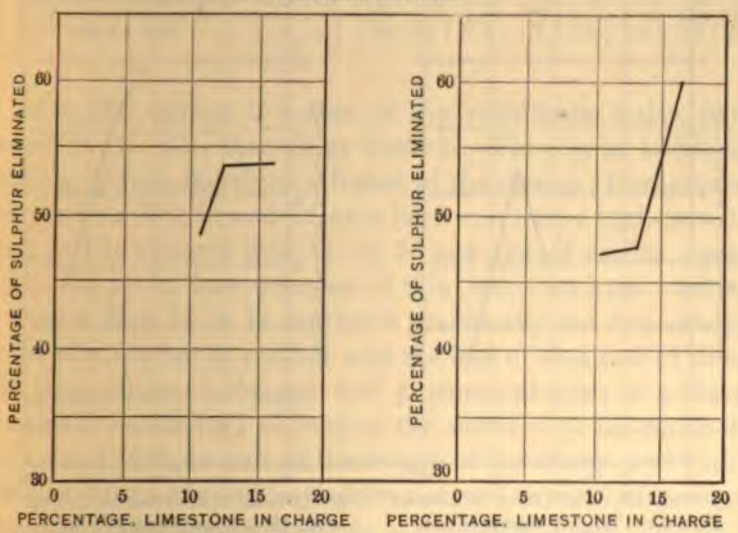

Fig. 13.—High-Pressure Blast.

Fig. 15.—Low-Pressure Blast.

Figs. 12 to 15.—Residual and Eliminated Sulphur.

with 20 per cent. The loss in silver, represented by curve No. 11, follows that of the lead; it increases with additions of limestone to 16 per cent. and then falls, at first gradually, and later more quickly. As limestone forming 24 per cent. of the weight of the charge is the permissible limit, if, when blown,

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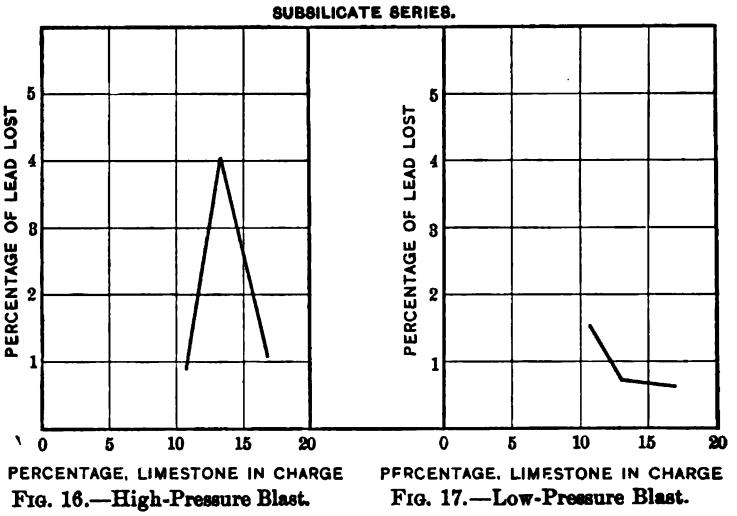

FIG. 16.—High-Pressure Blast. FIG. 17.—Low-Pressure Blast.

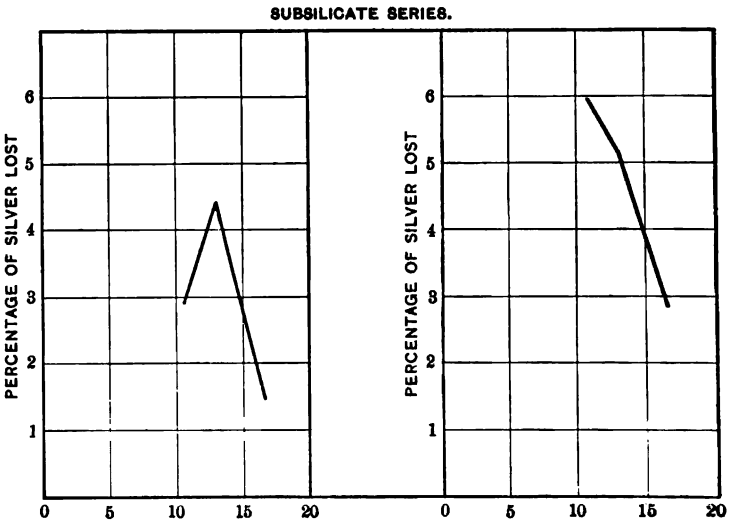

FIG. 18.—High-Pressure Blast. FIG. 19.—Low-Pressure Blast.

FIGS. 16 to 19.—LOSS OF LEAD AND SILVER.

it is to be satisfactorily slagged, it may be inferred that, for practical purposes, the loss in silver increases with the additions of limestone. If curve No. 11 is compared with curves Nos. 9 and 10, a similarity between the two will be seen, indicating that the loss in silver increases with the elimination of sulphur and of lead.

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TABLE III.—*Subsilicate Series.*

	Number of charge	8	9	9a	10	11	12	13	13a
Charge.	Ore, grams	1,000	1,000	1,000	1,000	1,000	1,000	1,000	1,000
	Limestone, grams	120	150	150	200	120	150	200	200
	Silica, grams	....	....	....	....	....	....	....	....
	Water, per cent	5	5	5	5	5	5	5	5
	Limestone, per cent. in charge	10.7	13	13	16.6	10.7	13	16.6	16.6
	Silicate degree, O base : O acid	1.03:1	1.06:1	1.06:1	1.12:1	1.03:1	1.06:1	1.12:1	1.12:1
	Time taken feeding, min.	5.5	5.5	7.5	8	5	4.75	6	9.5
	Duration of run, min.	18	14	14	17	16	17	17	18
Blast.	Pressure while charging, in. water	2.25	2.5	2	2.5	2	2	2	2
	Pressure, maximum, in. water	8.25	10	6	10	2	2	2	2
Product.	Weight, grams	1,012	1,041	1,085	1 087	1,020	1,085	1,080	1,107
	Slagged part, per cent	98	96.35	98.1	86.9	98.1	98.1	98.8	95.6
	Pulverulent part, per cent	2	3.65	1.9	13.1	1.9	1.9	1.2	4.4
Sulphur.	Sulphur in ore-charge, grams	105.8	105.8	105.8	105.8	105.8	105.8	105.8	105.8
	Sulphur in ore-charge, per cent.	9.45	9.20	9.20	8.82	9.44	9.20	8.81	8.81
	Sulphur in product, per cent.	5.4	4.7	4.85	4.45	5.47	5.35	3.76	3.91
	Sulphur in product, grams.	54.6	48.9	50.2	48.4	55.8	55.4	40.6	43.3
	Sulphur eliminated, per cent.	48.4	53.8	52.5	53.9	47.4	47.6	61.6	59.1
Lead.	Lead in ore-charge, grams	547	547	547	547	547	547	547	547
	Lead in ore-charge, per cent.	48.8	47.6	47.6	45.6	48.8	47.6	45.6	45.6
	Lead in product, per cent.	53.6	50.45	54.5	49.85	52.8	52.5	50.44	52.25
	Lead in product, grams.	542	525	564	541	539	543	545	578
	Lead lost, per cent.	0.9	4.0	gain	1.1	1.5	0.7	0.6	gain
Silver.	Silver in ore-charge, grams	1.35	1.35	1.35	1.35	1.35	1.35	1.35	1.35
	Silver in product, grams.	1.31	1.35	1.29	1.33	1.27	1.28	1.31	1.35
	Silver lost, per cent.	2.96	0.0	4.4	1.5	5.9	5.2	2.9	2.9

Table III., giving the data of the subsilicate series, is arranged in the same manner as Table II. There is an additional heading, giving the silicate-degree of the charge; furthermore, charges Nos. 8, 9, 9*a* and 10 have been run with a high-pressure blast, and in charges Nos. 11, 12, 13 and 13*a*, of similar composition, the initial blast-pressure of 2 in. has been kept constant.

Curves Nos. 12 to 19 represent graphically the data relating to the elimination of sulphur and the loss of lead and of silver. The proportions of slagged and pulverulent parts of a charge remained satisfactory as long as the addition of limestone did not exceed 16.6 per cent. of the weight of the charge, and with it the percentage of sulphur was not reduced below 8.81 per cent. and that of lead below 45.6 per cent.; beyond these limits the blown charge was not sufficiently coherent to be suited for a blast-furnace charge. Curves Nos. 12 to 15 show that the residual sulphur (3.76 to 5.47 per cent.) and the sulphur eliminated (47.4 to 61.6 per cent.) decrease and increase respectively with the addition of limestone until this forms 13 per cent. of the charge. Comparing curve No. 12 with No. 14, and No. 13 with No. 15, shows that the oxidation of sulphur proceeds more

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quickly with a high than with a low pressure, but that it is not so effective; the percentages of residual sulphur are greater and the percentages of elimination smaller. Comparing curves Nos. 14 and 15 of the low-blast subsilicate series with the corresponding curves, Nos. 8 and 9, of the singulo-silicate series, it is found that the percentages of residual sulphur and elimi-

TABLE IV.—*Medium-Size Singulo-Silicate Charges.*

	Number of charge	16	17
Charge.	Ore, kg	20	20
	Limestone, kg	4	5.4
	Silica, kg	1.04	1.35
	Water, kg	0.75	1.33
	Limestone, per cent. of charge	16	20
	Time taken feeding, min.	15	15
	Duration of run, min	45	45
Blast.	Pressure while charging, in. water	5	7
	Pressure, maximum, in. water	12	14½
Product.	Weight of product, kg	23.2	23.6
Sulphur.	Sulphur in ore-charge, grams	2,116	2,116
	Sulphur in ore-charge, per cent	8.45	7.91
	Sulphur in product, per cent	3.85	3.55
	Sulphur in product, grams	893	838
	Sulphur eliminated, grams	1,223	1,278
	Sulphur eliminated, per cent	57.8	60.4
Lead.	Lead in ore-charge, grams	10,940	10,940
	Lead in ore-charge, per cent	43.7	40.9
	Lead in product, per cent	48.6	47.15
	Lead in product, grams	11,275	11,127
	Lead lost, grams	gain	gain
	Lead lost, per cent	gain	gain
Silver.	Silver in ore-charge, grams	27.00	27.00
	Silver in product, grams	26.58	26.55
	Silver lost, grams	0.42	0.45
	Silver lost, per cent	1.56	1.67

nated sulphur are respectively lower and higher in the singulo-silicate than in the sub-silicate series. The loss in lead, curves Nos. 16 and 17, is greater with high- than with low-pressure blast. This is due, no doubt, to the greater prevalence of blow-holes with the former. It is greater in the singulo-silicate than in the sub-silicate series. The probable explanation is to be found in the presence of a larger amount of slag-forming material for a given quantity of lead. The loss in silver with the

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subsilicate series is shown in curves Nos. 18 and 19. It is seen to be proportional to the loss of lead, as was the case with curves Nos. 10 and 11 in the singulo-silicate series.

Table IV. gives the data of the two tests made with singulo-silicate charges using 20 kg. of ore. In order to compare the effect of the size of charge, the leading data are brought together in Table V.

TABLE V.—*Comparison of Large and Small Singulo-Silicate Charges.*

Size of Charge.	Large.		Small.	
Limestone added, per cent	20	27	20	26.7
Sulphur in product, per cent	3.85	3.55	3.75	3.56
Sulphur eliminated, per cent	57.8	60.4	60.8	59.9
Lead lost, per cent	gain	gain	4.55	0.0
Silver lost, per cent	1.56	1.67	7.0	6.4

Passing over the irregularities in the percentage of lead lost, Table V. shows one decided difference—viz., that the loss in silver is very much greater with a small than with a large charge. There is no reason apparent why this should be the case. Without a larger number of tests to substantiate the greater loss, this, for the present, must be considered as accidental.

IV. CONCLUSION.

The above experiments were carried on with a single ore, and the results find direct application only to this ore or to one very similar to it. Nevertheless, they have a more general application; they point to the conclusions:

1. That in lime-roasting a siliceous galena-concentrate low in blende, charges containing a wide range of lime and silica, and little iron, can be successfully blown.

2. That a singulo-silicate charge, with limestone equal to 16 per cent. of the weight of the ore, gives most satisfactory results as regards the physical condition of the product, the elimination of sulphur, and the loss of lead and silver.

3. That the same is true with a subsilicate charge.

4. That a low-pressure blast is a better desulphurizer and causes less loss than a high-pressure blast.

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The Constitution of Ferro-Cuprous Sulphides.

BY H. O. HOFMAN, W. S. CAYPLESS AND E. E. HARRINGTON, MASSACHUSETTS INSTITUTE OF TECHNOLOGY, BOSTON, MASS.

(New York Meeting, April, 1907.)

I. INTRODUCTION.

AT the Lake Superior meeting, September, 1904, Messrs. A. Gibb and R. C. Philp presented a paper entitled "The Constitution of Mattes Produced in Copper-Smelting,"[1] in which they concluded that ferrous and cuprous sulphides formed the chemical compound, 5 Cu_2S, FeS. They arrived at this inference from the observed fact that a matte containing Cu_2S, 90.4, and FeS, 9.6 per cent. (or, in round figures, Cu_2S, 90, and FeS, 10 per cent.), did not contain any metallic copper, while one with more than 10 per cent. of FeS readily dissolved copper, and with less than this amount retained copper in mechanical suspension. Although they determined the melting-points of five mattes containing from 32.6 to 80.1 per cent. of copper, they did not draw a freezing-point curve, as the necessary apparatus was not available. A microscopical examination of 11 polished samples gave them a grayish-black field of very uniform appearance, except that five of the specimens showed the presence of metallic copper.

The object of the present investigation was to supplement the above paper by drawing the freezing-point curve of the ferrous sulphide-cuprous sulphide series, and to see how the constitution could be further revealed by microscopical work. The results obtained lead to conclusions very different from those of Gibb and Philp. While their paper bears the date 1904, it was published only in 1906.[2] In the meantime, Bolles[3] had presented his paper, "The Concentration of Gold and Silver in Iron-Bottoms." The raw material which formed the basis of the research was a matte containing Fe, 61.68, and Cu, 11.20

[1] *Trans.*, xxxvi., 665 to 680 (1906).
[2] *Trans.*, xxxvi., 665 (1906).
[3] *Trans.*, xxxv., 666 to 695 (1905).

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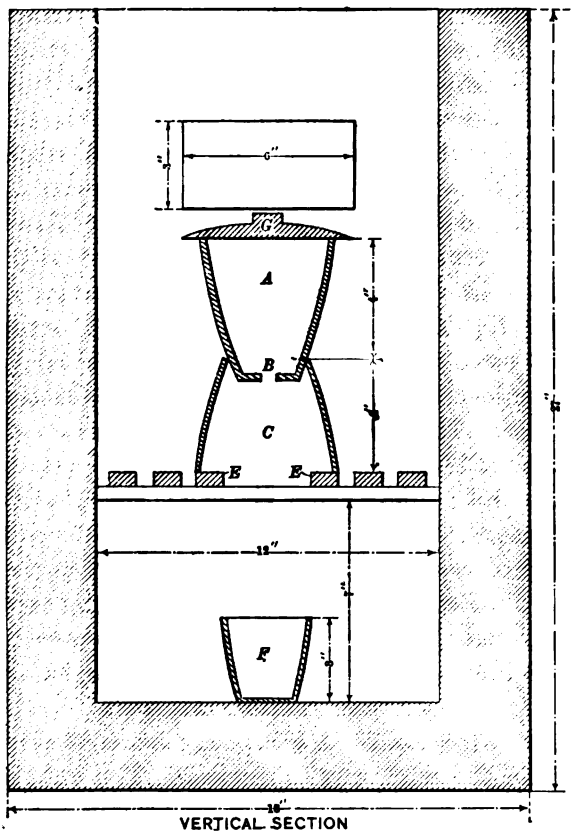
per cent., produced by a smelting-plant, and therefore not pure; nevertheless, the microscopic results foreshadow some of the facts found in the present work.

II. Preparation of Ferrous and Cuprous Sulphides.

The two sulphides were prepared by heating metallic iron and copper with stick-sulphur. Fig. 1, showing the apparatus used, represents a vertical section through the pot-furnace.



VERTICAL SECTION

Fig. 1.—Pot-Furnace.

This contains a size K Hessian crucible, *A*, with a hole in the bottom, *B*, 0.5 in. in diameter, supported by a second crucible, *C*, of the same size, with bottom cut off for the length of 1 in. and inverted. The support rests upon grate-bars, *E*, which, keeping the coal and ashes away from the inclosed space, makes it possible to collect clean sulphide in the size B Denver crucible, *F*, placed in the ash-pit.

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Soft iron wire, No. 14 B. & S. gauge, cut into 6-in. lengths, was placed in the perforated crucible. When this had been brought to an orange-red, the cover, *G*, was removed. From 10 to 15 g. of stick-sulphur, crushed to hazelnut-size, was added, and the cover replaced. Iron and sulphur combined readily, and the iron sulphide, melting at a low temperature, flowed through the hole in the crucible and was caught in the receiver. When the flow of iron sulphide ceased, fresh additions of sulphur were made. If too much sulphur was added at one time, it simply ran out through the bottom of the crucible without uniting with the iron.

In order to obtain a uniform compound in which all the iron was combined with the sulphur, the first iron sulphide was broken into pieces of hazelnut-size and placed in a graphite crucible with additional sulphur; powdered charcoal was added and the crucible covered to exclude the air; and the charge heated in the pot-furnace until the sulphide had become entirely liquefied. The crucible was then removed and its content allowed to cool slowly. The chemical analysis gave Fe, 64.57 per cent., while FeS consists of Fe, 63.54, and S, 36.46 per cent. The excess of iron present is due to the fact observed by Le Chatelier and Ziegler,[4] that ferrous sulphide heated above its melting-point, air being excluded, loses sulphur, and that the loss in sulphur increases with the temperature. A photomicrograph is shown in Fig. 4. The ferrous sulphide was full of cavities and fissures, caused by evolution of gas and by shrinkage in cooling. It was therefore difficult to find parts giving a smooth surface for microscopical examination.

Pure copper wire, No. 15 B. & S. gauge, cut into 6-in. lengths, was treated in the same way as the iron. The temperature of the copper had to be higher than that of the iron for the sulphur to combine with it. The cake of re-melted sulphide showed, when broken, radial fracture-planes forming elongated particles with surfaces resembling hard specular iron-ore in color and luster. The chemical analysis gave Cu, 78.94 per cent., while the theoretical percentage of copper in cuprous sul-

[4] *Bulletin de la Société d'Encouragement pour l'Industrie Nationale*, ciii., p. 368 (1902); *Metallographist*, vol. vi., p. 19 (1903).

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phide is 79.68. A photomicrograph of this sulphide is shown in Fig. 14.

III. Determination of the Freezing-Point Curve.

The apparatus used at first for fusing mixtures of ferrous and cuprous sulphides was an electric resistance furnace. It consisted of a porous fire-clay cup, 3.25 in. in diameter and 7 in. high, wound with 9 ft. of platinum wire of No. 26 B. & S. gauge, a cylinder of galvanized iron 10.5 in. in diameter and 12 in. high, and magnesia packing. The winding was calculated to give, with 8 amperes, a temperature of from 1,200° to 1,300° C. On account of the care and the long time (about 45 min.) it took to bring the furnace to the required temperatures, it was abandoned and replaced by a small No. 40 Fletcher gas-furnace, 5 in. in diameter, which was easily heated to 1,200° C. in 5 min. The rapid cooling characteristic of this type of furnace was obviated by inclosing it in a 1-in. covering of magnesia and using a cover of the same material with a hole in the center. The cooling was sufficiently slow to accentuate retardation-points of the fused mixtures held in No. 2 graphite crucibles.

Temperatures were measured with the Le Chatelier thermo-electric pyrometer. The wires were insulated from one another by a double-bored clay tube, and the hot-junction protected from the corrosive action of melted matte by a Berlin porcelain tube, 0.5 in. in diameter and 10 in. long, one end of which had been closed by fusing with an oxy-hydrogen gas blow-pipe. The current produced by heating the thermo-junction was measured with a Sullivan mirror galvanometer. The instrument was read by the movement of a reflected ray of light on a ground glass-plate, on the lower side of which a strip of millimeter plotting-paper had been pasted. On calibrating, the instrument gave, with the cold-junction at 28° C., a deflection of 280 mm. for the copper point and one of 160 mm. for the aluminum point, or 120 scale-divisions corresponded to 427° C., or one division to about 3.5° C. In Fig. 2, representing the arrangement of apparatus, *A* is a table; *B*, Fletcher furnace; *C*, stand; *D*, handle of pyrometer; *E*, cold-junction thermometer; *F*, Sullivan mirror galvanometer; *G*, pyra-

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midal wooden hood ($\frac{1}{8}$-in. stock); *H*, incandescent electric lamp; *I* and *J*, supports.

The charges used at first weighed 20 g., but as these cooled too quickly, the quantity was doubled, which was found to be sufficient to mark distinctly the first retardations while cooling. In other experiments that are contemplated the weight of a charge will be increased to 100 g. to bring out more points for the freezing of the eutectic. In making up the charges, the necessary quantities of ferrous and cuprous sulphides were weighed out, sulphur was added, and the whole mixed and covered with a layer of fine charcoal.

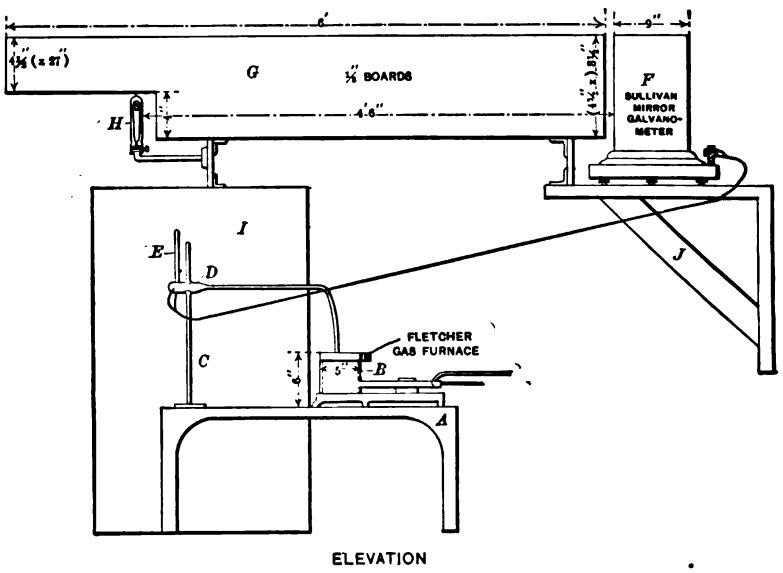

FIG. 2.—ARRANGEMENT OF MELTING-APPARATUS.

When a charge was fused, the heated porcelain protecting-tube was inserted, to be followed by the thermo-couple. When this indicated the temperature of fused mixture, the gas-burner was shut off and the crucible and content allowed to cool slowly in the usual way—*i.e.*, without stirring the charge. Readings were taken by noting the time in seconds it took the ray of light to drop half a division on the scale. It was observed that when the gas-burner had been removed, the temperature continued to rise as much as 25° C. before it began to fall. Similarly, when heat was applied to a charge which had

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been cooling, the cooling continued for some time before the temperature started again to rise.

In allowing the charge to cool without stirring, the retardations often covered quite a range of temperature and masked the real point of solidification. If the charge was stirred, it solidified more uniformly and gave a more marked freezing-point.

In samples of matte with high percentages of cuprous sulphide the solidification proceeded more quickly and satisfactorily than when the percentages of ferrous sulphide were high. The readings taken with stirring are recorded in Table I.:

TABLE I.—*Record of Freezing-Points of Iron-Copper Sulphides.*

FeS.	Cu_2S.	Observed Retardations.
Per Cent.	Per Cent.	Degrees Centigrade.
100	0	978
95	5	920
90	10	884, 886
85	15	885
80	20	850, 858, 859, 859, 859, 879
75	25	855, 856
70	30	854, 858
65	35	850, 858, 860, 865
60	40	862, 913, 917, 919
50	50	978
40	60	1,028
30	70	1,053
20	80	1,090, 1,094
10	90	1,123, 1,126, 1,129
0	100	1,152

The retardation-points in matte sometimes show marked differences. This is probably due to the facts, 1, that a matte does not solidify as suddenly as does a metal or an alloy composed of metals; 2, that a porcelain tube is attacked by matte, especially when rich in ferrous sulphide; 3, that matte has the property of dissolving small amounts of slag; and 4, that the limit of accuracy of a pyrometer-reading was 3.5° C. Second retardations in the mattes were noticed only three times: at 862° C., with the matte containing FeS, 60, and Cu_2S, 40 per cent.; at 850° C., with the mattes containing FeS, 65, and Cu_2S, 35 per cent.; and FeS, 80, and Cu_2S, 20 per cent. The point 850° C. was sufficiently marked to accept this, at least provisionally, as the freezing-point of the eutectic. The freezing-point curve drawn in Fig. 3 has a V-form characteristic for

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eutectiferous alloys. The sudden peculiar deflection of the curve from its general course between FeS, 60, and Cu_2S, 40 per cent., and FeS, 65, and Cu_2S, 35 per cent., seems to be borne out by the abrupt change of structure shown in the photomicrographs, Figs. 10 and 11. The ferrous-sulphide and cuprous-sulphide branches are drawn to meet at the point representing the composition, FeS, 86, and Cu_2S, 14 per cent., as the microscopical examination showed that the eutectic point lay a little beyond FeS, 85 per cent. Similar reasons justified dotting the eutectic line from FeS, 90 per cent., to the eutectic

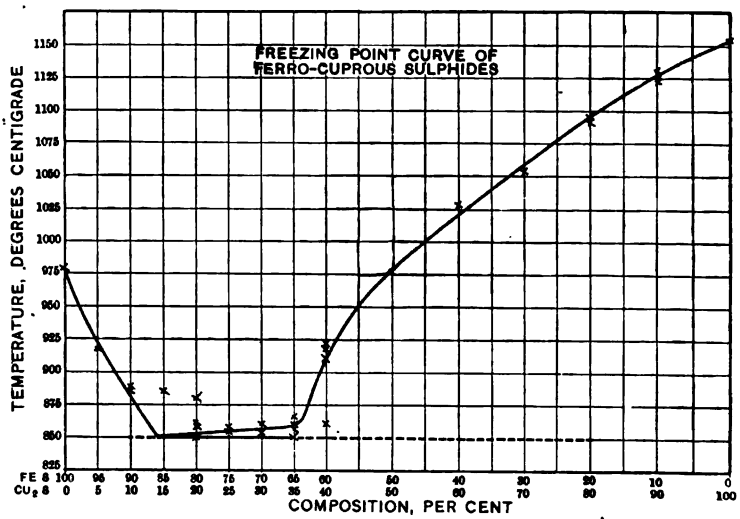

FIG. 3.—FREEZING-POINT CURVE OF FERRO-CUPROUS SULPHIDES.

point, and from FeS, 65, and Cu_2S, 35 per cent., to FeS, 20, and Cu_2S, 80 per cent.

IV. MICROSCOPICAL EXAMINATION.

The samples were polished on a Sauveur machine in the usual way with coarse and fine emery, and with rouge; the polished surfaces were then cleaned and burnished on a disk covered with wetted broadcloth. Samples rich in cuprous sulphide were easier to polish than those rich in ferrous sulphide, since the latter were filled with pits, which was not the case with the former. Attempts at etching a polished surface with silver nitrate to attack the cuprous sulphide, and with potassium-copper chloride to attack the ferrous sulphide, did not help to develop

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the structure; on the contrary, they blurred what was distinct before. The samples were therefore microphotographed without having been etched, but a deep-orange screen was used to furnish the desired contrast on the negative between the blue rays of cuprous sulphide, the fawn-colored rays of ferrous sulphide, and the purplish-drab of one component of the eutectic. The time of exposure to the 110-volt arc-light, using Cramer's instantaneous isochromatic plates, was 1.25 min. with samples containing Cu_2S, from 100 to 50 per cent., while 1.5 min. was found to be necessary with the others. The magnification throughout was 300 diameters.

1. *FeS*, 100, *and* Cu_2S, 0 *per cent.*, Fig. 4, shows two constituents, white areas of ferrous sulphide and a gray eutectic resembling the impression made by a thumb-print. Under the microscope ferrous sulphide is fawn-colored. The eutectic consists of plates of ferrous sulphide and a component of a purplish-drab color. The black spots are cavities. Le Chatelier and Ziegler[5] distinguished three constituents, yellow areas of ferrous sulphide, white brilliant particles of metallic iron, and a eutectic composed of yellow sulphide and a mixture of iron oxide and metallic iron, with the former strongly predominating. The white specks were not visible in our sample, and immersion in a cupric sulphate solution showed only here and there a slight tracery of metallic copper.

2. *FeS*, 95, *and* Cu_2S, 5 *per cent.*—The structure is the same as that of pure ferrous sulphide (Sample No. 1), with this difference, that the plates making up the eutectic are larger and slightly deeper in color.

3. *FeS*, 90, *and* Cu_2S, 10 *per cent.*, Fig. 5, shows the effect of a larger addition of cuprous sulphide, which was to reduce in size the fawn-colored areas of ferrous sulphide and the purplish-drab colored eutectic. The excess ferrous sulphide is more interwoven with the eutectic than before, and the cavities and fissures are also smaller in number and in size.

4. *FeS*, 85, *and* Cu_2S, 15 *per cent.*, Fig. 6, presents a general appearance which differs from that of Sample No. 3. Nearly the entire field is covered by the eutectic, showing the components as before; there is no indication of excess ferrous sul-

[5] *Bulletin de la Société d'Encouragement pour l'Industrie Nationale*, ciii., p. 368 (1902); *Metallographist*, vol. vi., p. 19 (1903).

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FIG. 4.

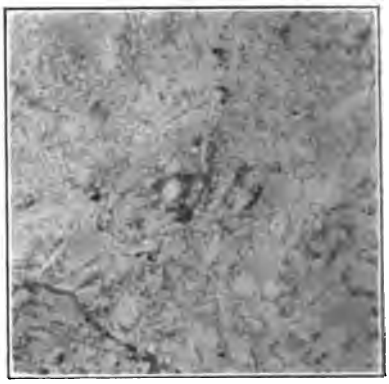

FIG. 5.

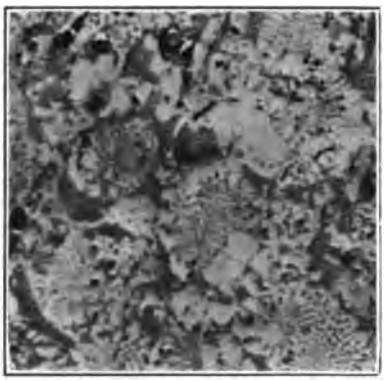

FIG. 6.

FIG. 7.

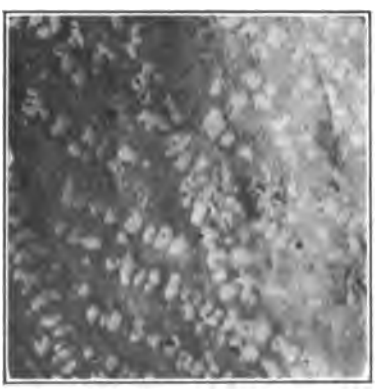

FIG. 8.

FIG. 9.

FIGS. 4 TO 9.—MICROPHOTOGRAPHS OF FERRO-CUPROUS SULPHIDES, MAGNIFIED 300 DIAMETERS.

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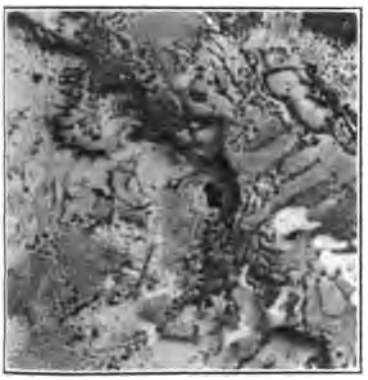

FIG. 10.

FIG. 11.

FIG. 12.

FIG. 13.

FIG. 14.

FIGS. 10 TO 14.—MICROPHOTOGRAPHS OF FERRO-CUPROUS SULPHIDES, MAGNIFIED 300 DIAMETERS.

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phide, which has disappeared in the eutectic. The dark members of the eutectic have become prominent, they are often bunched together, and show here and there a strawberry-like surface. For the first time there appear a few sky-blue areas of cuprous sulphide. In the print they are darker than the dark component of the eutectic, have no connection with it, and show no structure whatever. The appearance of sky-blue cuprous sulphide proves that the eutectic point of the alloy has been passed, but only slightly, as, with the exception of a few small areas, the whole field is made up of eutectic. This was the reason for drawing the ferrous- and cuprous-sulphide branches so that they met at FeS, 86 per cent.

5. *FeS*, 80, *and* Cu_2S, 20 *per cent.*, Fig. 7, is the print made from the clearest negative that could be obtained from the sample, which appeared hazy under the microscope. The sky-blue parts of cuprous sulphide, light-colored in the print, have become more numerous. The structure of the eutectic is fine; while obscured in the print, it is visible under the microscope.

6. *FeS*, 75, *and* Cu_2S, 25 *per cent.*, Fig. 8. The print of this sample resembles very much that of sample No. 5. The light-colored particles of cuprous sulphide are smaller, more numerous and better defined; the eutectic hardly reveals any structure.

7. *FeS*, 70, *and* Cu_2S, 30 *per cent.*, Fig. 9, shows that the microscopical aspect has completely changed. There is a sky-blue area of cuprous sulphide, with particles of a fawn to purplish-drab color disseminated through it; the eutectic islands are fawn-colored where the structure is indistinct, and a more-purplish drab where it is readily seen. Specks of metallic copper, white in the print, make their first appearance. The resulting enrichment of the matte in ferrous sulphide was not taken into account.

8. *FeS*, 65, *and* Cu_2S, 35 *per cent.*, Fig. 10, seems to show that the free development of components has been hampered, which gives the photograph a different aspect from that of Sample No. 7, although the constituents remain the same. The dark structureless parts are cuprous sulphide; the eutectic structure is very marked; white particles of copper are less scarce and larger.

9. *FeS*, 60, *and* Cu_2S, 40 *per cent.*, Fig. 11, shows that the grouping of the constituent parts is different from any of those

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of Samples Nos. 1 to 8. In the picture the dark knob-like structureless parts are sky-blue cuprous sulphide; these appear to be enclosed by light-colored rings, which seem to be surrounded by a dark mass showing here and there some structure. Under the microscope the blue is clearly the sea in which float eutectic fawn-colored islands, in the centers of which has been assembled the other constituent with its purplish-drab color. This sudden change of structure, shown in Figs. 10 and 11, denotes a decided change in the reciprocal solubility of the two sulphides, and is brought out clearly in the freezing-point curve.

10. *FeS*, 50, *and* Cu_2S, 50 *per cent.*, Fig. 12, has the same general character as Fig. 11; the constituents are, however, larger, and the purplish-drab component of the eutectic presents more structure.

11. *FeS*, 40, *and* Cu_2S, 60 *per cent.*, Fig. 13, presents a background of blue cuprous sulphide, having large and small eutectic islands of a fawn color, showing very little structure. The white patches are metallic copper, which reaches here its maximum and decreases quickly in the subsequent samples.

12. *FeS*, 30, *and* Cu_2S, 70 *per cent.* The photograph of this material differs too little from that of Fig. 13 to warrant reproduction.

13. *FeS*, 20, *and* Cu_2S, 80 *per cent.*; *FeS*, 10, *and* Cu_2S, 90 *per cent.*; *FeS*, 0, *and* Cu_2S, 100 *per cent.* Fig. 14 shows that all these samples have a plain blue field without any structure whatever; all contain small particles of metallic copper. This disappearance of all structure with the composition, FeS, 20, and Cu_2S, 80 per cent., justifies drawing the eutectic line up to this point.

V. Conclusions.

In summarizing the results, the conclusions arrived at are:

1. Ferrous and cuprous sulphides form no chemical, but a eutectiferous compound.

2. The structure of the eutectic of ferrous and cuprous sulphides becomes merged in that of the pure ferrous sulphide and cannot be distinguished from it.

3. The limited reciprocal solubility of the two sulphides diminishes along the cuprous-sulphide branch of the curve, slowly at first, then more quickly; solubility is complete beyond the alloy, FeS, 20, and Cu_2S, 80 per cent.

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Relative Elimination of Iron, Sulphur, and Arsenic in Bessemerizing Copper-Mattes.

BY E. P. MATHEWSON, ANACONDA, MONT.

(New York Meeting, April, 1907.)

THE experiments described in this paper were made at the Washoe Reduction Works, Anaconda, Mont., for the purpose of determining the relative speed of elimination of the iron, sulphur and arsenic during the process of bessemerizing copper-mattes.

The samples were taken from a horizontal "barrel" converter, 8 ft. in diameter and 12 ft. 6 in. long, having an average charge of 9 tons of matte, and blown with a 16-lb. air-pressure. All samples were taken with the blast on, turning the converter from its normal blowing-position to nearly vertical, so that a conical mold, with a long handle attached to a horizontal rod, could be introduced from a platform above the converter into the mouth of the converter and a sample taken. The difficulties of taking a sample with the blast on are apparent, and the irregularities of the lines in Figs. 1, 2 and 3 can be readily accounted for by the amount of slag in the samples. On the whole, the plotting of the results gives comparatively uniform lines, considering the conditions.

The samples were taken by the regular sampling-force of the works at 10-min. intervals, as shown in Tables I., II. and III., which also give the chemical analysis of the samples so taken. The test shown in Table I. was made with a new converter; that of Table II. with a converter that had made three charges and had been cleaned out; that of Table III. with a converter that had blown one charge to white metal, and had then been "washed" out and used for the test.

In plotting these results, it was necessary to take, as a basis, the percentage of copper in the samples. This basis does not exactly represent the true condition, since the volume is constantly decreasing and a certain quantity of copper passes into the slag, but it is as near as can be obtained in practice. Take, for instance, in Table I., the sample No. 5, which shows: Cu, 59.9;

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Fe, 13.7; S, 21.95 per cent., by actual analysis; then the plotted percentages are the percentages that 13.7 and 21.95 per cent. are of 59.9 per cent.; or 22.87 and 36.64 per cent., respectively. In the charts it will be noticed that the 10-min. intervals are plotted as the abscissæ, and the percentages of Cu, Fe, S, in 5-per cent. intervals, and arsenic in 0.05-per cent. intervals, as the ordinates.

Table IV. shows the percentages of iron and sulphur eliminated during the first period, or to that point immediately after the last skimming.

TABLE IV.—*Quantities of Iron and Sulphur Eliminated in Bessemerizing Copper-Mattes.*

Test.	Copper in Matte. Per Cent.	Iron Eliminated. in First Period. Per Cent.	Sulphur Eliminated in First Period. Per Cent.
April 12, 1906, . . .	40.68	98	60
March 7, 1906, . . .	46.08	96	53
April 5, 1906, . . .	54.81	97	44

Apparently the major portion of the arsenic that is eliminated is driven off during the first 30 minutes.

The analyses of these samples were made by the regular force under the supervision of Mr. H. N. Thomson, chief chemist; the plotting by Mr. H. R. Burg, statistician, and the compilation by William Wraith, assistant superintendent, all of the Washoe Reduction Works.

Tables I., II. and III., and the illustrations, Figs. 1, 2 and 3, are given in the following pages. Each table is placed on one page and its corresponding chart on the other, so that a ready comparison may be made between the statistical data and the diagrammatic representation.

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TABLE II.—*Test with Stall No.* 8, *Converter No.* 16, *April* 5, 1906.

	Time.		Composition.						
	Hr.	Min.	Cu.	Ag.	Au.	Insol.	Fe.	S.	As.
			Per Cent.	Oz.	Oz.	Per Cent.	Per Cent.	Per Cent.	Per Cent.
2.22 p.m., charged Nos. 1 and 2 blast matte			54.81	39.30	0.17	0.30	18.00	23.00	0.076
2.28 p.m., started to blow.									
2.38 p.m., sample No. 1 blowing		10	56.26	39.80	0.17	0.44	17.40	20.30	0.065
2.48 p.m., sample No. 2 blowing		20	38.25	26.90	0.13	4.04	30.10	14.20	0.042
2.58 p.m., sample No. 3 blowing		30	64.71	46.50	0.22	0.80	10.40	20.10	0.040
3.08 p.m., sample No. 4 blowing		40	68.86	49.90	0.23	0.68	7.40	19.50	0.040
3.18 p.m., sample No. 5 blowing		50	72.72	53.40	0.24	0.28	4.20	18.20	0.042
3.19 p.m., stopped blowing 3 min. to skim.									
3.22 p.m., blowing resumed.									
3.28 p.m., sample No. 6 blowing	1	00	78.29	57.50	0.27	0.56	1.60	19.50	0.042
3.28 p.m., stopped blowing, skimmed 2 min.									
3.30 p.m., blowing resumed.									
3.38 p.m., sample No. 7 blowing	1	10	79.49	58.90	0.28	0.36	0.50	18.60	0.042
3.48 p.m., sample No. 8 blowing	1	20	78.27	55.90	0.30	2.04	0.90	16.40	0.042
3.58 p.m., sample No. 9 blowing	1	30	79.63	52.70	0.10	2.80	1.00	14.90	0.042
4.08 p.m., sample No. 10 blowing	1	40	82.85	49.50	0.12	1.34	0.80	13.10	0.034
4.18 p.m., sample No. 11 blowing	1	50	98.46	90.50	0.54	0.058	0.019	0.73	0.053
4.28 p.m., sample No. 12 blowing	2	0	98.75	76.30	0.37	0.020	0.012	0.53	0.049
4.30 p.m., blow finished, copper			99.26	85.70	0.37	0.022	0.004	trace	0.052
Gross time	2	02							
Net time	1	57							

No punching on this charge. Sample taken each ten minutes from beginning of blow until finished.

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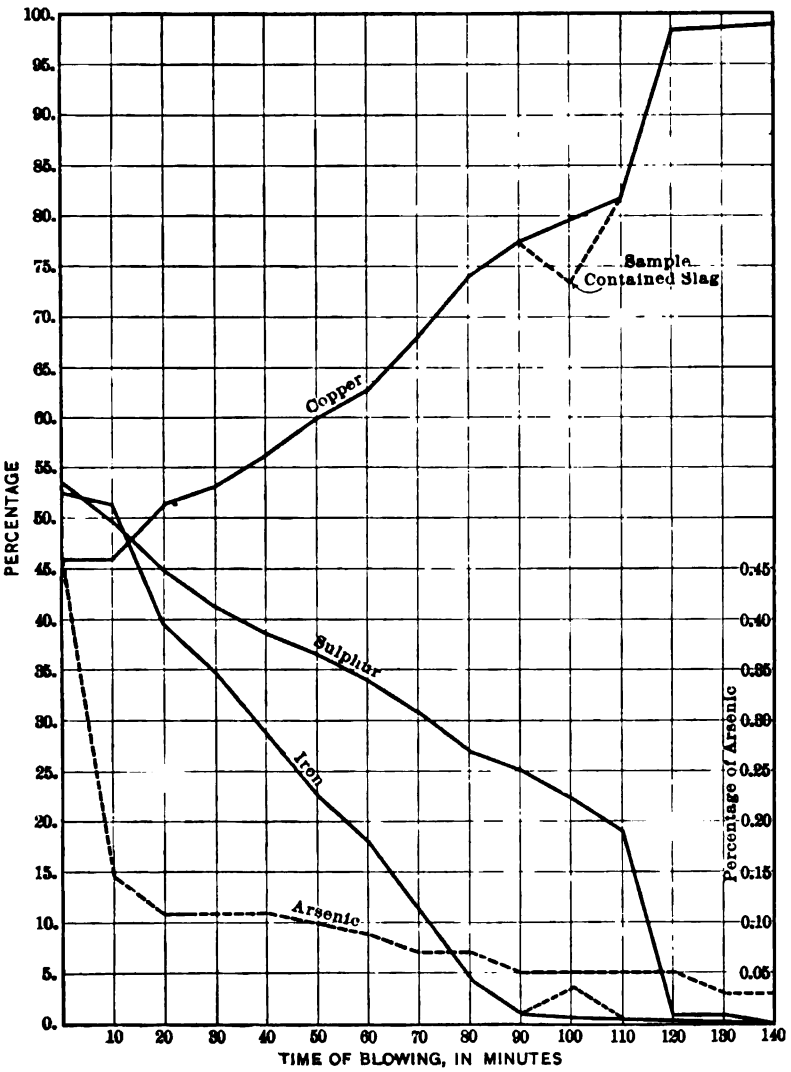

FIG. 1.—CHART OF TABLE I.

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TABLE II.—*Test with Stall No.* 8, *Converter No.* 16, *April* 5, 1906.

	Time.		Composition.						
	Hr.	Min.	Cu.	Ag.	Au.	Insol.	Fe.	S.	As.
			Per Cent.	Oz.	Oz.	Per Cent.	Per Cent.	Per Cent.	Per Cent.
2.22 p.m., charged Nos. 1 and 2 blast matte			54.81	39.30	0.17	0.30	18.00	23.00	0.076
2.28 p.m., started to blow.									
2.38 p.m., sample No. 1 blowing		10	56.26	39.80	0.17	0.44	17.40	20.30	0.065
2.48 p.m., sample No. 2 blowing		20	38.25	26.90	0.13	4.04	30.10	14.20	0.042
2.58 p.m., sample No. 3 blowing		30	64.71	46.50	0.22	0.80	10.40	20.10	0.040
3.08 p.m., sample No. 4 blowing		40	68.86	49.90	0.23	0.68	7.40	19.50	0.040
3.18 p.m., sample No. 5 blowing		50	72.72	53.40	0.24	0.28	4.20	18.20	0.042
3.19 p.m., stopped blowing 3 min. to skim.									
3.22 p.m., blowing resumed.									
3.28 p.m., sample No. 6 blowing	1	00	78.29	57.50	0.27	0.56	1.60	19.50	0.042
3.28 p.m., stopped blowing, skimmed 2 min.									
3.30 p.m., blowing resumed.									
3.38 p.m., sample No. 7 blowing	1	10	79.49	58.90	0.28	0.36	0.50	18.60	0.042
3.48 p.m., sample No. 8 blowing	1	20	78.27	55.90	0.30	2.04	0.90	16.40	0.042
3.58 p.m., sample No. 9 blowing	1	30	79.63	52.70	0.10	2.80	1.00	14.90	0.042
4.08 p.m., sample No. 10 blowing	1	40	82.85	49.50	0.12	1.34	0.80	13.10	0.034
4.18 p.m., sample No. 11 blowing	1	50	98.46	90.50	0.54	0.058	0.019	0.73	0.053
4.28 p.m., sample No. 12 blowing	2	0	98.75	76.30	0.37	0.020	0.012	0.53	0.049
4.30 p.m., blow finished, copper			99.26	85.70	0.37	0.022	0.004	trace	0.052
Gross time	2	02							
Net time	1	57							

No punching on this charge. Sample taken each ten minutes from beginning of blow until finished.

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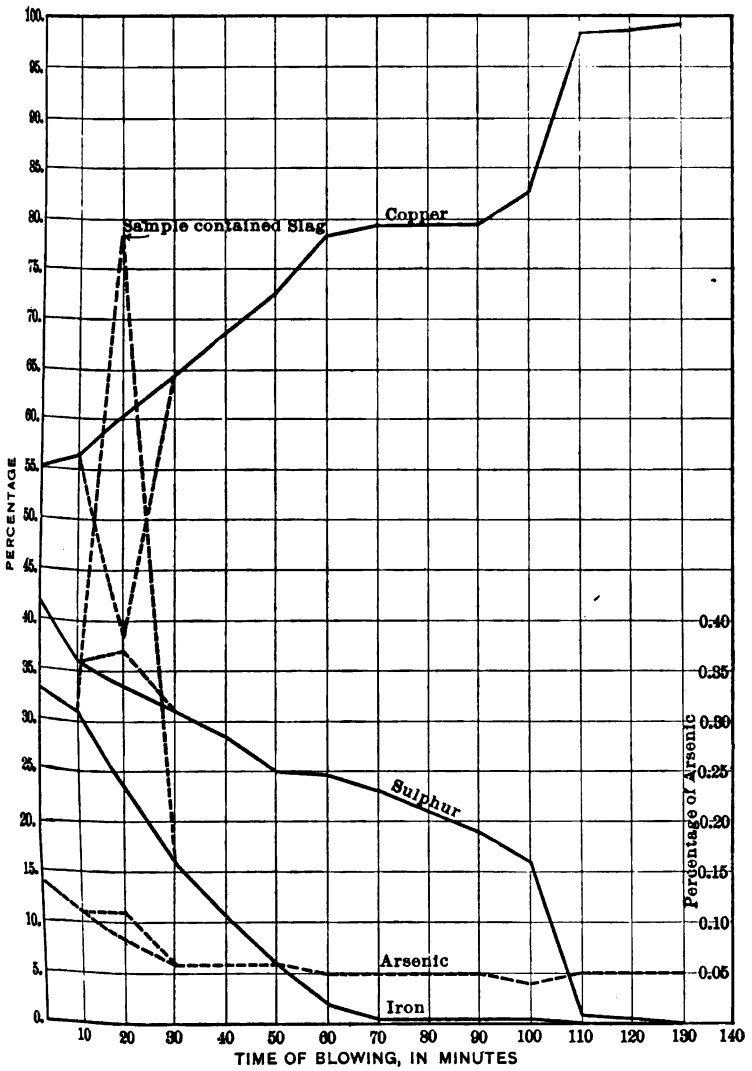

FIG. 2.—CHART OF TABLE II.

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Roasting of the Argentiferous Cobalt-Nickel Arsenides of Temiskaming, Ontario, Canada.

BY HENRY M. HOWE, LL.D., WILLIAM CAMPBELL, PH.D., AND CYRIL W. KNIGHT, B.SC.*

(New York Meeting, April, 1907.)

THIS paper gives the results of an investigation of the behavior of the argentiferous cobalt-nickel arsenides of Temiskaming, Ontario, in roasting, made in the metallurgical laboratories of the School of Mines of Columbia University in the City of New York. The ore was kindly given by the owners of the La Rose and Trethewey properties at Cobalt, Ontario, and Mr E. J. Hall, Tutor in Assaying in Columbia University, has helped us much.

I. OBJECT OF THE INVESTIGATION.

The object of the investigation was to learn:

1. The temperature at which the arsenic is most rapidly expelled;

2. The thoroughness with which it is expelled by prolonged roasting at this temperature;

3. The effect of adding charcoal (*a*) near the end of the roast and (*b*) at the beginning of the roast.

II. NATURE OF THE ORE.

The important ores mined in the Temiskaming deposits are: native silver—with small amounts of dyscrasite (Ag_6Sb), argentite (Ag_2S), pyrargyrite (Ag_3SbS_3)—smaltite ($CoAs_2$), chloanthite ($NiAs_2$) and niccolite (NiAs). Mispickel (FeAsS) and cobaltite (CoAsS) also occur in smaller quantities. The average composition of the ore shipped from this district for the first six months of 1905 was: silver, from 4.1 to 4.8 per cent.; cobalt, from 6.9 to 8.2 per cent.; nickel, from 3 to 4.7 per cent.; and arsenic, from 30.9 to 34.6 per cent. The ore which we treated consists chiefly of smaltite. In our laboratory-work

* Respectively Professor, Instructor and Student of Metallurgy in the School of Mines of Columbia University in the City of New York.

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the temperature was kept below the melting-point of silver (960° C.) in order to prevent loss of that metal, though our preliminary experiments showed that these ores do not frit or clog at this or even at a somewhat higher temperature.

The Temiskaming ores contain little gold, only $0.40 per ton in case of the ores which we treated.

III. Sampling and Assaying.

About 43 lb. of the ore, in lumps about 3-in. cubes, was crushed so as to pass through a sieve of 20 meshes to the linear inch. In this crushing we caught and separated particles of metallic silver which represented about 75 oz. of silver to the ton of ore, or about 11 per cent. of its total value. Some of these particles were about 0.25 in. in diameter.

For assaying, a lot of 37 assay-tons was next separated from this crushed product by means of a "split" sampler, and then ground so as to pass a sieve of 100 meshes to the linear inch. In doing this a second lot of metallic silver particles, representing 117 oz. per ton of ore, was separated.

The results of 12 crucible-assays and 9 scorification-assays were:

Metallic silver by the crucible-process, ...	477 oz. per ton of ore.
Metallic silver by the scorification-process, . . .	497 oz. per ton of ore.
Add metallic silver separated in crushing, . . .	75 oz. per ton of ore.
Add metallic silver separated in grinding, . . .	117 oz. per ton of ore.
Total silver-content, ...	689 oz. per ton of ore.

The ore contains 56 per cent. of arsenic.

For the silver-assay the following quantities were used: For scorification, 0.2 A. T. of ore was roughly divided into halves, each of which was scorified with 60 g. of lead and 1 g. of borax-glass. The resulting two beads were weighed together. For the crucible process, 0.5 A. T. of ore, 2.5 A. T. of lead oxide, ½ A. T. of soda, ½ A. T. of borax-glass, and 4.5 g. of argol were used.

IV. The Roasting.

The roasting was done in an American Gas Furnace Company's oven No. 2 (Fig. 1), 27 in. long and 20 in. wide inside.

The ore was held in shallow iron pans resting on the hearth of this furnace, and lined with 0.5 in. of fire-brick.

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The temperature was measured by means of a Le Chatelier pyrometer. The thermo-couple, *C*, was protected from arsenic and other fumes by a porcelain tube, *D*, and was placed immediately above the ore, *A*. It entered through a circular hole *E*, in the back of the furnace, and was connected with a Keiser and Schmidt galvanometer, standardized by means of the melting-points of zinc, aluminum and copper. The temperature was recorded and the ore rabbled every 10 min. In no. of the roasts was any fritting or clogging of the ore noticed.

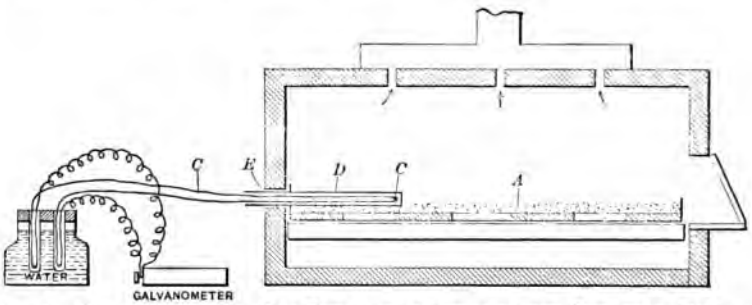

FIG. 1.—SECTION OF THE GAS-FURNACE IN WHICH THE ORE WAS ROASTED, SHOWING THE ARRANGEMENT OF THERMO-COUPLE FOR INDICATING THE TEMPERATURE.

V. AT WHAT TEMPERATURE IS THE ARSENIC EXPELLED MOST RAPIDLY?

Roast No. 1.—In this test, 3.5 lb. of the ore, crushed to pass a 20-mesh sieve, was placed in the furnace, the temperature of which had previously been raised to 490° C. The temperature was then gradually raised at the rate of about 120° C. per hour, until it finally reached its highest point, 870° C. Samples were taken with the usual precautions, and their arsenic was determined by fusing with sodium peroxide, neutralizing with acetic acid and sodium hydroxide (using phenolphthalein as an indicator), precipitating as silver arsenate with silver nitrate, and titrating with ammonium thiocyanate.[1]

The results of the roast are given in Table I., and graphically in Fig. 2.

Though the arsenic escaped pretty rapidly at first, yet towards the end of the second hour its escape was almost com-

[1] Miller's *Quantitative Analysis for Mining Engineers*, p. 114.

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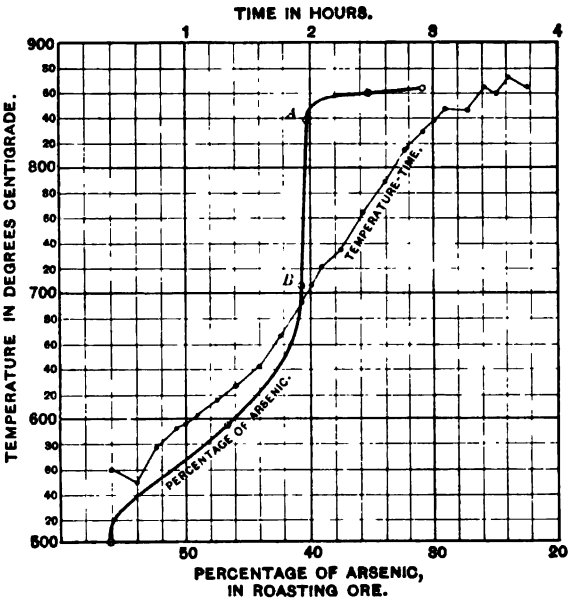

FIG. 2.—DIAGRAMMATIC RECORD OF ROAST NO. 1. (See Table I.)

TABLE I.—*Record of Roast No. 1. Gradually Rising Temperature.*

Temperature of Roast.	Time from the Beginning.	Quantity of Arsenic in Ore at Different Stages of the Roast.
Degrees Centigrade.	Hr. Min.	Per Cent.
490	0 : 00	55.9
559	0 : 25	
547	0 : 35	
577	0 : 45	
591	0 : 55	
596	1 : 00	46.6
601	1 : 05	
611	1 : 15	
624	1 : 25	
641	1 : 35	
666	1 : 45	
691	1 : 55	
706	2 : 00	40.7
721	2 : 05	
735	2 : 15	
763	2 : 25	
788	2 : 35	
814	2 : 45	
828	2 : 55	
838	3 : 00	40.3
848	3 : 05	
844	3 : 15	
864	3 : 25	
858	3 : 30	35.4
872	3 : 35	
863	3 : 45	31.0

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pletely arrested, in spite of the continued regular rise in temperature, as shown by the nearly vertical part, *A B*, of the arsenic-line in the figure. But when the temperature reached 840°, the expulsion of arsenic again became rapid, as is shown by the sharp bend of the arsenic-curve to the right at the point *A*. These results tend to prove that the behavior of smaltite resembles that of pyrite, of which the first atom of sulphur is removed at a much lower temperature than the second.

Conclusions.—1. That 15 per cent. of arsenic (per 100 of

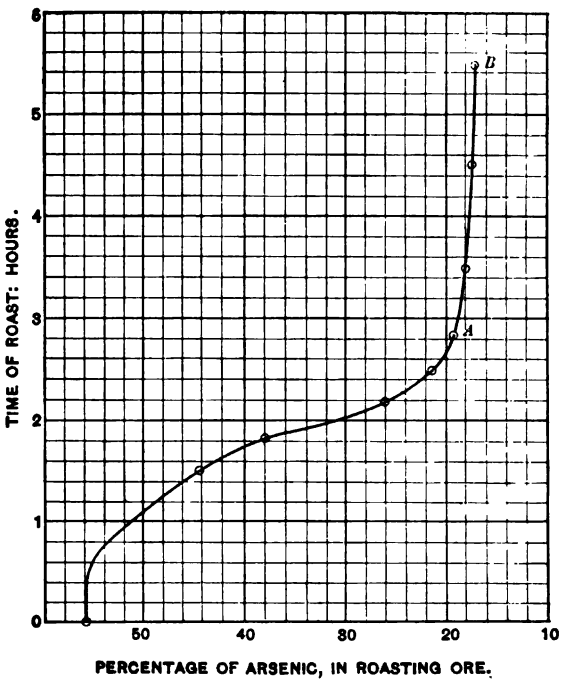

FIG. 3.—DIAGRAMMATIC RECORD OF ROAST No. 2. (See Table II.)

ore)—*i.e.*, 27 per cent. of the total arsenic—is expelled below 700° C.

2. That the rest of the arsenic is not expelled until the temperature reaches about 840°, when rapid expulsion again sets in.

VI. How Thoroughly Can Arsenic be Expelled at 890° C.?

Roast No. 2.—In this roast about 3 lb. of ore, ground to pass a sieve of 20 meshes to the linear inch, was raised quickly to

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about 890° C. (a temperature a little above that at which Roast No. 1 had shown that arsenic is rapidly expelled), and held near that temperature for about four hours, with frequent rabbling.

As shown in Table II. and Fig. 3, the arsenic was expelled fairly rapidly until it had fallen to about 20 per cent., but thereafter very slowly.

TABLE II.—*Record of Roast No. 2. Temperature Held Near 890° C.*

Temperature of Roast. Room Temperature.	Time.	Quantity of Arsenic in Ore at Different Stages of the Roast.
Degrees Centigrade.	Hr. Min.	Per Cent.
463	0 : 35	55.9
533	0 : 45	
622	0 : 55	
693	1 : 05	
738	1 : 15	
795	1 : 25	
846	1 : 35	44.6
895	1 : 45	
895	1 : 55	38.0
909	2 : 05	
897	2 : 15	26.3
901	2 : 25	
886	2 : 35	21.7
883	2 : 45	
883	2 : 55	19.3
883	3 : 05	
885	3 : 15	
889	3 : 25	
883	3 : 35	18.3
891	3 : 45	
894	3 : 55	
899	4 : 05	
897	4 : 15	
902	4 : 25	
900	4 : 35	17.5
900	4 : 45	
898	4 : 55	
887	5 : 05	
883	5 : 15	
878	5 : 35	17.3

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VII. Does Charcoal Added after Roasting at 890° Cause Further Expulsion of Arsenic?

Table III.—*Record of Roast No. 3. Charcoal Added After Roasting Near 890° C.*

Temperature of Roast. Room Temperature.	Time from the Beginning.	Quantity of Arsenic in Ore at Different Stages of the Roast.
Degrees Centigrade.	Hr. Min.	Per Cent.
470	0 : 35	55.9
606	0 : 45	
660	0 : 55	
723	1 : 05	
815	1 : 15	
856	1 : 25	
873	1 : 35	
886	1 : 45	
872	1 : 55	
864	2 : 05	
870	2 : 15	
874	2 : 25	
888	2 : 35	
875	2 : 45	
897	2 : 55	
890	3 : 05	15.8
	Charcoal added here.	
882	3 : 20	
882	3 : 35	
894	3 : 50	
884	4 : 05	15.0

Roast No. 3.—In order to learn whether an addition of charcoal after long roasting between 840° and 890° C. (temperatures between which we had found that arsenic is expelled rapidly) causes further expulsion of arsenic by reducing the fixed arsenates to the volatile forms of arsenious acid and metallic arsenic, one of us roasted a third lot of ore at temperatures between 870° and 890° for 1.5 hr., and, without removing it from the furnace, he then stirred in 10 per cent. by weight of charcoal, ground so as to pass a sieve of 10 meshes to the linear inch, but not one of 20 meshes. The results are shown in Table III. If too coarse, charcoal disintegrates and scatters the ore, and if too fine it burns away too fast.

At the time of adding the charcoal, fumes of arsenic had ceased to be visible, but this addition caused a sudden evolution of dense fumes, which lasted for only a few minutes.

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The charcoal had little effect on the arsenic. Before its addition the ore contained 15.8 per cent. of arsenic, and 1 hr. later this had fallen only to 15 per cent.

In this roast, after the ore had been exposed to a temperature above 856° for 1 hr. 40 min., its arsenic-content had fallen to 15.8 per cent., whereas in Roast No. 2, after it had been exposed 2 hr. to temperatures above 846°, it still contained 18.3 per cent. of arsenic. This difference tends to show that unnoticed variations in conditions may materially influence the rate of expulsion, as is the case in many roasting-operations.

The fact that the arsenic was expelled in this roast, before the addition of the charcoal, more thoroughly than in any of the others, in spite of the very rapid raising of the temperature at the beginning, goes to show that the behavior of smaltite differs in an important way from that of pyrite, the temperature of which must be raised very carefully and slowly, lest the fritting or enameling of the outer surface of the individual particles prevent the free access of the air to their interior, and thus arrest the roast.

VIII. Does Charcoal Added at the Beginning of the Roast Increase the Expulsion of Arsenic?

Roast No. 4.—About 2 lb. of ore was mixed with 10 per cent. by weight of charcoal, raised to 880° in 2 hr., and held near that temperature for 1.75 hr. more, or a total of 3 hr. 45 min. The charcoal seems to have had little effect, because at the end of this time the ore still contained 17.5 per cent. of arsenic, or more than in Roast No. 3 after it had been above 856° for 1 hr. 40 min., and but little less than in Roast No. 2 after it had been above 795° for 2 hr. 10 min.; and in each of these latter cases the expulsion of arsenic was brought about without the use of charcoal.

These cases are here recapitulated:

Roast.	After Remaining Above	For	The Ore Still Contained
No. 2.	846° without charcoal.	2 hr.	18.3 per cent. of arsenic.
No. 3.	856° without charcoal.	1 hr. 40 min.	15.8 per cent. of arsenic.
No. 4.	880° with charcoal.	1 hr. 45 min.	17.5 per cent. of arsenic.

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IX. Does Finer Grinding Increase the Expulsion of Arsenic?

Roast No. 5.—In order to learn whether finer grinding would lead to further expulsion of arsenic by exposing the ore more fully to the air, the ore which had already undergone Roast No. 3 was re-ground so as to pass a sieve of 100 meshes to the linear inch, and re-roasted for 2 hr. 30 min. at about 880° C.; but this re-roasting caused no further expulsion of arsenic.

X. Summary of Results.

The following conclusions apply only to the particular ore here treated:

1. The percentage of silver as determined by the scorification-method is about 4 per cent. higher than as determined by the crucible-method.

2. The ore neither clogs nor frits at or even somewhat above 960°, the melting-point of silver.

3. The arsenic can be reduced from about 56 to 41 per cent., or by 15 per cent., by roasting below 700° C. (Roast No. 1, Table I., Fig. 2.)

4. It can be further reduced by about 24 per cent.—viz., to 17 per cent.—by roasting at temperatures above 840°, and in this higher range the arsenic is removed much faster than at lower temperatures. (Roast No. 2, Table II. and Fig. 3.)

5. Hence our inference that the behavior of smaltite in roasting is probably analogous to that of pyrite, which loses its first atom of sulphur much more readily than its second; yet, unlike pyrite, this ore may be raised suddenly to 800° without harm, because, unlike pyrite, it does not frit or enamel when thus suddenly heated, but remains open and porous, so that the air may penetrate it. (Roast No. 3.)

6. Charcoal, whether added at the beginning or towards the end of the roast, fails to increase the expulsion of arsenic. (Roasts Nos. 3 and 4.)

7. We doubt whether it will pay to reduce the arsenic-content below 20 per cent., even by roasting at temperatures above 890° C., because its further reduction is very slow.

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A Study in Refining and Overpoling Electrolytic Copper.

BY H. O. HOFMAN, R. HAYDEN AND H. B. HALLOWELL, MASSACHUSETTS INSTITUTE OF TECHNOLOGY, BOSTON, MASS.

(New York Meeting, April, 1907.)

I. INTRODUCTION.

THE object of refining copper in the reverberatory furnace is to obtain a metal which will have the highest attainable degree of malleability, ductility and electric conductivity, and present at the same time a level surface when it solidifies in the mold after casting. These desirable physical properties are governed by the character of the impurities and the forms in which they are present, by the amount of cuprous oxide retained by the copper, by the quantity of gas held in solid solution, by the casting-temperature, and by the thickness of the casting. The effects of impurities, of cuprous oxide and of gases upon the mechanical properties of copper have been studied by Hampe[1] in his classical paper "Contributions to the Metallurgy of Copper." The most recent research into the effect of metals upon the electrical conductivity of copper is that of Addicks.[2] The influence of cuprous oxide upon the electrical conductivity has been investigated by Walker[3] and Addicks.[4] The absorption of gases has received attention by Hampe,[5] Stahl[6] and Heyn.[7] The effects of casting-temperature have been noted by Stahl.[8]

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[1] *Zeitschrift für Berg-, Hütten-und Salinen-Wesen in Preussen*, vol. xxi., pp. 218 to 283 (1873); vol. xxii., pp. 93 to 138 (1874).

[2] *Journal of the Franklin Institute*, vol. clx., p. 425 (1905); *Trans.*, xxxvi., 18 to 27 (1906).

[3] *The Mineral Industry*, vol. vii., p. 248 (1898).

[4] *Transactions of the American Institute of Electrical Engineers*, vol. xxii., pp. 695 to 702 (1903); *Electrochemical Industry*, vol. i., pp. 580 to 583 (1902-03).

[5] *Op. cit.*, vol. xxi., p. 274 (1873); also *Chemiker Zeitung*, vol. xvii., p. 1692 (1893).

[6] *Ueber Raffination, Analyse und Eigenschaften des Kupfers*, Fieke, Altenau i. Harz (1886); also, *Berg- und Hüttenmännische Zeitung*, vol. xlviii., pp. 323, 324 (1889); vol. xlix., p. 399 (1890); vol. lii., p. 19 (1893); vol. lx., pp. 77 to 79 (1901).

[7] *Zeitschrift des Vereines deutscher Ingenieure*, vol. xliv., p. 508 (1900); *Metallographist*, vol. vi., p. 48 (1903); also, *Metallurgie*, vol. iii., p. 82 (1906).

[8] *Op. cit.*

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The present paper contains the results of two lines of investigations embodying (1) a study of the physical and chemical changes undergone by two charges of electrolytic copper at different plants while being refined in the reverberatory furnace, and (2) a study of overpoling electrolytic copper on eight tough-pitch and four furnace-overpoled samples from separate refineries. The physical changes considered were in appearance of surface, specific gravity, tensile strength, elongation and electric conductivity. The modifications in fracture and microstructure have already been studied by Hofman, Green and Yerxa,[9] and are therefore omitted. The chemical changes noted were confined to variations in the content of copper, iron, sulphur and oxygen, as the foreign substances contained in electrolytic copper are too small in amount to affect the present investigation. In the second part of the paper, the physical and chemical properties of samples of tough-pitch and of furnace-overpoled copper were studied; the tough-pitch samples were completely overpoled in crucibles and the ensuing properties of crucible-overpoled copper determined; lastly, crucible-overpoled copper was compared with native copper from Lake Superior.

II. Studies in Refining Electrolytic Copper.

1. *Refining-Charge No.* 1.

(*a*) *Samples.*—Twelve samples formed the basis of the first series of tests. They were taken from a refining-furnace of 100 tons capacity. Sample No. 1 was taken after melting down the electrolytic copper and skimming the slag. Sample No. 2, taken six hours later, represents set copper. During the following two and a quarter hours of poling, samples Nos. 3 to 11 were taken at 15-min. intervals; sample No. 11 is tough-pitch copper. The charge was then cast, with the exception of a small amount, which was overpoled until a cast bar upon cooling "threw a worm" or "spewed." A section of this bar formed sample No. 12. Samples Nos. 1 to 10, represented in Figs. 1 to 10, were 3 in. long and 1 in. square, being one-half of the test-bar usually cast during refining. All the surfaces are uneven until tough-pitch copper has approximately been

[9] *Trans.*, xxxiv., 671 to 695 (1904).

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reached with Fig. 10. Samples Nos. 1 to 5 show cavities; these disappear with sample No. 6, taken when poling had progressed for 45 min. Samples Nos. 11 and 12, representing tough-pitch and furnace-overpoled copper, are shown in cross-

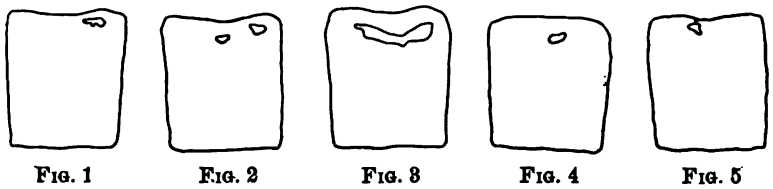

FIG. 1 FIG. 2 FIG. 3 FIG. 4 FIG. 5

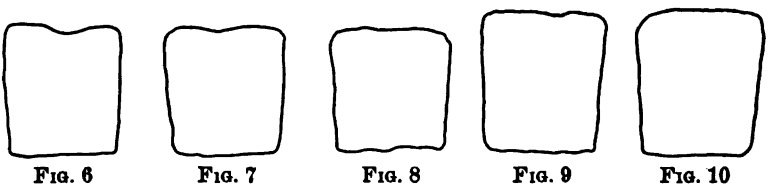

FIG. 6 FIG. 7 FIG. 8 FIG. 9 FIG. 10

FIGS. 1 TO 10.—SAMPLE-BARS OF REFINING-CHARGE NO. 1.

section in Figs. 11 and 12; their photographed surfaces are given in Figs. 13 and 14. The tough-pitch copper has the characteristic wrinkled level surface; the furnace-overpoled copper

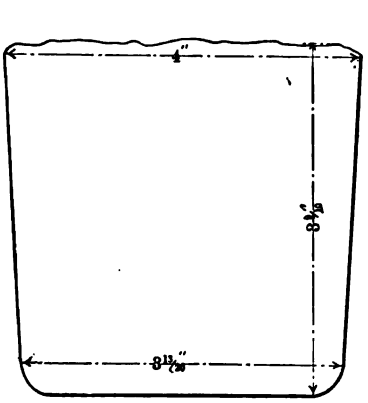

FIG. 11.—CROSS-SECTION OF TOUGH-PITCH WIRE-BAR.

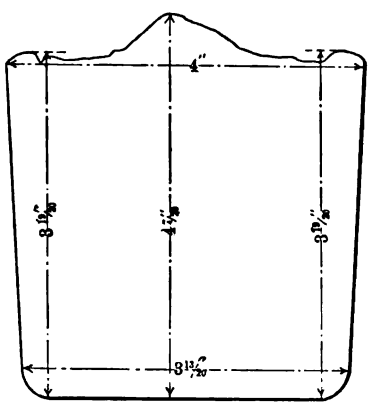

FIG. 12.—CROSS-SECTION OF FURNACE-OVERPOLED WIRE-BAR.

has a rough surface, a large ridge at the center and two small ones at the sides.

(*b*) *Specific Gravity.*—The figures for specific gravity, given in Table I., were calculated from the data obtained in the electrical tests by the formula, spec. gr. $= \frac{w}{Al}$, in which w is the

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weight of the wire sample in grams, A the area in square centimeters, and l the length in centimeters.

TABLE I.—*Specific Gravity.*

Sample No.....	1.	2.	3.	4.	5.	6.	7.	8.	9.	10.	11.	12.
Specific gravity	8.642	8.116	8.466	8.606	8.662	8.787	8.854	8.878	8.889	8.900	8.906	8.824

(*c*) *Tensile Strength and Elongation.*—On account of the form of the samples, the mechanical tests had to be made with specimens drawn into wire. For this purpose, pieces 0.5 by 1 in. and 2.5 in. long were cut from the specimens in such a way as to leave the surfaces of the originals intact. The pieces were drawn to 0.04 in. in diameter, corresponding to No. 18 B. & S. gauge, and then annealed together by a Connecticut brass manufacturing company. Difficulties in drawing were encountered only with samples Nos. 1 to 4, the last drawings of which had to be made by hand. The wires obtained varied in length from 15 to 35 ft., with the exception of that from sample No. 2 (set copper), which gave a length of only 7 ft. The tests were made with a Fairbanks wire-testing machine, the wires used being about 2 ft. long. The averages of the results are assembled in Table II.

TABLE II.—*Tensile Strength and Elongation.*

Sample No.	Tensile Strength.	Elongation in 8 In.
	Lb. per Sq. In.	Per Cent.
1	34,400	29.0
2	29,600	26.6
3	33,520	23.2
4	31,580	8.6
5	34,070	29.7
6	33,320	32.7
7	31,650	33.7
8	31,320	26.1
9	31,900	33.9
10	31,200	30.1
11	30,970	32.2
12	31,650	31.6

(*d*) *Electric Conductivity.*—The tests were made with a Wheatstone bridge, using wire-lengths of about 5 ft. The averages of the results are given in Table III.

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FIG. 13.—TOUGH-PITCH WIRE-BAR.

FIG. 14.—FURNACE-OVERPOLED WIRE-BAR.

FIG. 19.

FIG. 20.

FIGS. 19 AND 20.—SURFACE OF FURNACE-OVERPOLED WIRE-BAR, SHOWING WORM.

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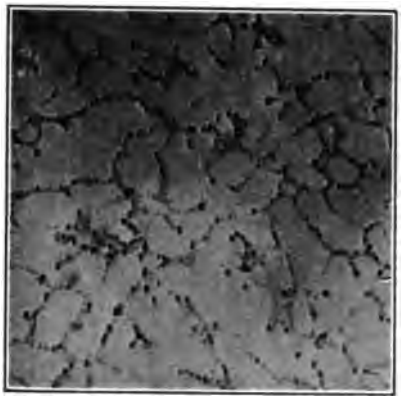

FIG. 25.—TOUGH-PITCH ELECTROLYTIC COPPER.

FIG. 26.—FURNACE-OVERPOLED ELECTROLYTIC COPPER.

FIG. 27.—TOUGH-PITCH ELECTROLYTIC COPPER.

FIG. 28.—FURNACE-OVERPOLED ELECTROLYTIC COPPER.

FIG. 29.—CRUCIBLE-OVERPOLED COPPER.

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TABLE III.—*Electric Conductivity.*

Sample No.....	1.	2.	3.	4.	5.	6.	7.	8.	9.	10.	11.	12.
Electric conductivity.	98.16	86.61	92.72	94.34	96.27	99.72	101.23	101.15	100.65	101.60	101.36	100.75

(*e*) *Chemical Changes.*—Only the chemical changes relating to copper, iron, sulphur and oxygen were considered. Copper, iron and sulphur were determined by chemical analysis, oxygen by planimetric measurement. All analytical work was carried through according to the methods perfected by G. L. Heath and given in his paper,[10] Methods for the Complete Analysis of Refined Copper. The wires from the physical tests formed the analytical material. Two separate samples had to be weighed out for the determinations, one for copper and iron, and one for sulphur, as the copper was deposited electrolytically from a sulphuric acid solution. Iron was precipitated from the sulphate solution after this had been freed from copper. In the planimetric measurement of oxygen from enlarged photomicrographs, the mode of procedure given by Hofman, Green and Yerxa[11] was followed. The averages of the results are given in Table IV.

TABLE IV.—*Analytical Results.*

Sample No.....	1.	2.	3.	4.	5.	6.	7.	8.	9.	10.	11.	12.	13.
Cu (+ Ag).....	99.22	98.12	98.50	99.25	99.52	99.61	99.66	99.70	99.74	99.82	99.87	99.70	.....
FeO.....	0.121	0.086	0.022	0.017	0.016	0.017	0.018	0.016	0.017	0.019	0.020	0.067	0.035
SO_3	.....	0.030	0.080	0.029	0.032	0.029	0.028	0.030	0.027	0.027	0.028	0.021	tr.
O	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	0.0119	0.0794	none
Measured planimetrically.													

(*f*) *Discussion of Data.*—In order to bring out the results more clearly than is convenient in the detached tables, and thus facilitate a review, all the data have been assembled and represented graphically on a single sheet in Fig. 15. Their discussion is confined for the present to samples Nos. 1 to 11, inclusive; samples Nos. 12 and 13, dealing with overpoled copper, will be taken up later.

The copper (plus silver) content, which at the start (sample No. 1) was 99.22 per cent., is seen to fall to 98.12 per cent.,

[10] *Journal of the American Chemical Society*, vol. xxvii., pp. 308 to 318 (1905).

[11] *Trans.*, xxxiv., 671 to 695 (1904).

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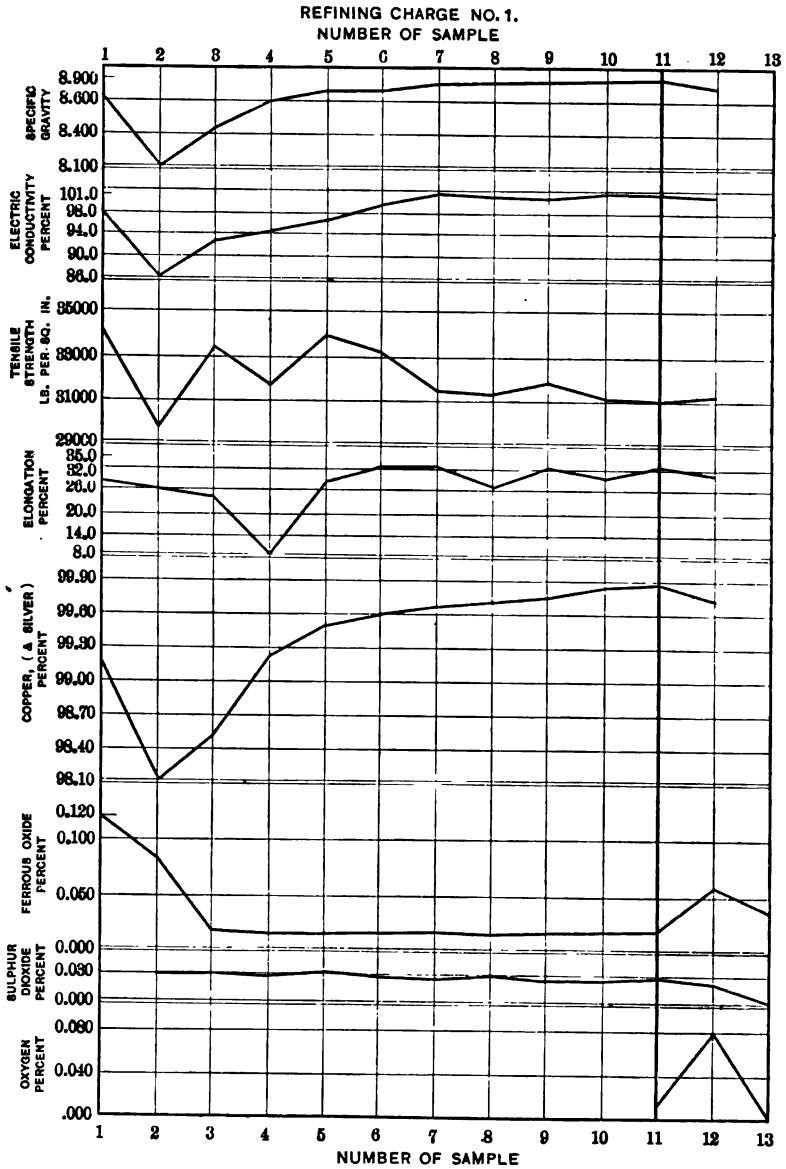

FIG. 15.—PHYSICAL AND CHEMICAL CHANGES OF ELECTROLYTIC COPPER IN REFINING.

when, after oxidation for 6 hr., the stage of set copper (sample No. 2) has been reached. During the first hour of poling the percentage of copper rises quickly, reaching 99.61 per cent. with

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sample No. 6, then more slowly, attaining the maximum of 99.87 per cent. with tough-pitch copper (sample No. 11).

Ferrous oxide, usually lower than sulphur, is here higher. It shows a peculiar behavior. It would have been expected that the original 0.121 per cent. would have been slagged off completely during the six hours of oxidizing fusion, but it was reduced with set copper to 0.086 per cent., and only then brought to the minimum of 0.022 per cent. by the first quarter-hour of poling, to remain practically unchanged. The only explanation that suggests itself is that iron was taken up from the pipe through which air was forced into the copper during the oxidizing stage, and that this was quickly expelled when the pipe had been withdrawn and the pole inserted.

While the sulphur-content is high for electrolytic copper, it remains practically constant at 0.030 per cent., the extreme figures being 0.027 and 0.032 per cent. It appears, then, that with electrolytic copper no sulphur is eliminated during the refining operation.

Oxygen determinations were confined to sample No. 11, tough-pitch copper, which contained 0.0119 per cent.

The specific-gravity curve shows the same general trend as that of the copper-content, as was to be expected; starting with 8.642, it reaches the minimum of 8.116 with set copper, and the maximum of 8.906 with tough-pitch copper.

Electric conductivity gives a curve resembling those of copper-content and specific gravity. The electric conductivity, 98.16 per cent. with sample No. 1, reaches its minimum of 86.61 per cent. with set copper (sample No. 2), and then rises quickly to 101.23 per cent. with sample No. 7, 1.25 hr. after poling had been started, and remains approximately at that figure for the additional one hour of poling necessary to reach the tough-pitch state. While electric conductivity has become a very important test for judging the physical properties of copper, the curve shows that, at least in the present case, the conductivity-test did not tell the whole story, even though the copper under consideration was a high-grade metal.

The tensile strength of 34,400 lb. of sample No. 1 shows a gradual decrease to 30,970 lb. with tough-pitch copper (sample No. 11). Fluctuations in the curve between samples Nos. 3

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and 7 are caused by the difficulty in adjusting the wires, brittle at this stage, in the jaws of the machine.

The elongation increases as the poling progresses: the irregularities are due to the same causes as those of the variations in the tensile-strength tests. Starting with 29 per cent., it reaches a minimum of 8.6 per cent. half an hour after poling has begun, and a maximum of 33.9 per cent. half an hour before the stage of tough-pitch copper.

2. *Refining-Charge No. 2.*

(*a*) *Samples.*—Fifteen samples, taken from a refining-furnace of 120 tons capacity, were examined to study the changes that took place during the operation. Six samples, Nos. 13 to 18, were small ingots, $4\frac{1}{8}$ in. long by $2\frac{1}{4}$ in. wide at top and $1\frac{5}{16}$ in. wide at bottom by $1\frac{1}{8}$ in. thick. Sample No. 13 represents set copper, sample No. 14 was taken after the first pole had been withdrawn, sample No. 15 after the two subsequent poles had been used up, sample No. 16 after poles 4 and 5 had been taken out, sample No. 17 after poles 6 and 7 had been removed, and sample No. 18 after the copper had reached the tough-pitch stage. The specimen for microscopical examination was taken from the center of a cross-sectional piece cut off from the end of a bar; the material for chemical analysis was obtained by boring five holes $\frac{3}{8}$ in. in diameter into the bottom of a bar, a hole penetrating one-half. The six samples, Nos. 13a to 18a, were duplicates of Nos. 13 to 18, cast into the form of a nail, $5\frac{3}{8}$ in. long and $\frac{3}{8}$ in. in diameter at the top, and 0.5 in. at the bottom. The lower half of a nail was cut off to be drawn into wire for the mechanical and electrical tests. The wires were drawn to No. 18 B. & S. gauge at the works of the American Steel & Wire Co., Worcester, Mass. The drawn wires were annealed together. Difficulties similar to those with the brittle specimens of refining-charge No. 1 were also encountered here. The three samples, Nos. 19 to 21, are sections of full-size wire-bars of furnace-overpoled copper; their contours, shown in Figs. 16 to 18, represent typical crowned surfaces. Figs. 19 and 20 are photographs of the surfaces of two of the samples in which the worm thrown was very pronounced.

(*b*) *Specific Gravity.*—The specimens polished for microscopical examination served for the determinations of the specific

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gravity, made in the usual way by weighing in air and in water with the necessary precautions. The data obtained are given in Table V.

TABLE V.—*Specific Gravity.*

Sample No.....	13.	14.	15.	16.	17.	18.	19.	20.	21.
Specific gravity.	8.23	8.36	8.47	8.63	8.61	8.69	8.12	8.24	8.58

(*c*) *Tensile Strength and Elongation.*—The mechanical tests were carried out in the same manner and with the same ma-

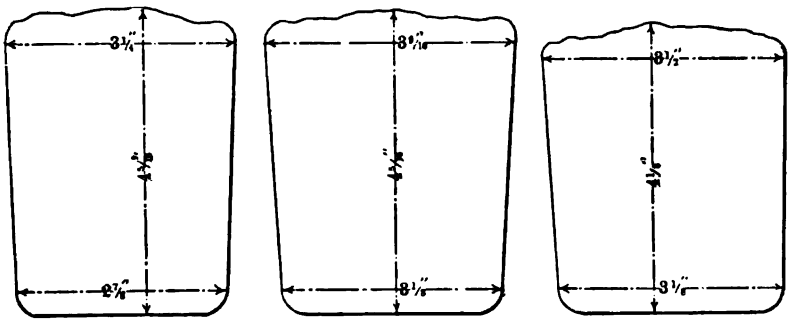

FIG. 16. FIG. 17. FIG. 18.

FIGS. 16 TO 18.—CROSS-SECTIONS OF FURNACE-OVERPOLED WIRE-BAR.

chine as those of refining-charge No. 1. The results are given in Table VI.

TABLE VI.—*Tensile Strength and Elongation.*

Sample No.....	13.	14.	15.	16.	17.	18.	19.	20.	21.
Tensile strength, lb. per sq. in.	39,130	39,020	38,560	37,520	36,630	37,020	35,460	35,720	35,400
Elongation in 10 in., per cent.	34.6	31.2	30.1	31.0	34.2	35.1	34.5	28.2	37.5

(*d*) *Electric Conductivity.*—The tests for electric conductivity were made at the Worcester plant of the American Steel & Wire Co. The figures are assembled in Table VII.

TABLE VII.—*Electric Conductivity.*

Sample No.....	13.	14.	15.	16.	17.	18.	19.	20.	21.
Electric conductivity, per cent.	98.2	98.3	98.7	99.6	99.2	99.6	99.8	99.5	100.3

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(*e*) *Chemical Changes.*—In addition to following up the changes which take place in the content of copper, iron and sulphur of the metal-bath during refining, planimetric measurements of oxygen were made of all the samples. The results are brought together in Table VIII.

TABLE VIII.—*Chemical Changes.*

Sample No.....	13.	14.	15.	16.	17.	18.	19.	20.	21.
Cu (+ Ag).....	99.53	99.67	99.79	99.91	99.93	99.94	.....	.....	.....
FeO.....	0.0063	0.0059	0.0063	0.0057	0.0059	0.0063	.....	.....	.....
SO_2.....	0.048	0.042	0.045	0.039	0.042	0.043	.....	.....	.....
O.....	0.211	0.203	0.171	0.073	0.056	0.044	0.033	0.0063	0.018

(*f*) *Discussion of Data.*—The data obtained in examining samples Nos. 13 to 21 are plotted in Fig. 21. The distance on the abscissa between samples Nos. 13 (set copper) and 18 (tough-pitch copper) has been made approximately the same as that between samples Nos. 2 and 11 of Fig.15, which stand for the same limits in the refining of a charge.

The copper (and silver) content, 99.53 per cent., of sample No. 13, is the lowest of the series, as the start was made with set copper. In poling, it rises quickly at first to 99.91 per cent., sample No. 16, and then only very slowly reaches the maximum of 99.94 per cent. with tough-pitch copper, sample No. 18.

The determinations of ferrous oxide gave a range of 0.0057 and 0.0063 per cent., and the curve rises and falls within it without any regularity whatever. This indicates that the iron is not distributed evenly throughout the mass of the bath, and that it is not diminished in amount during the period of poling.

The variations in sulphur dioxide, 0.039 to 0.048 per cent., are greater than those of ferrous oxide, although the largest difference does not exceed 0.009 per cent. Some sulphur is expelled by poling, as set copper contains 0.048 per cent. sulphur dioxide and tough-pitch copper 0.043 per cent., but the amount is insignificant.

The oxygen curve forms an interesting inverse to that of the copper-content. The 0.211 per cent. oxygen of set copper decreased quickly with the poling until sample No. 16, with 0.073 per cent. oxygen, was taken, and then slowly, being reduced only 0.029 per cent. when the metal had been brought

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to the tough-pitch stage, ready to be cast. The figure of 0.211 per cent. oxygen (= 1.43 per cent. cuprous oxide) for set copper is very low; and would seem to show that with this charge

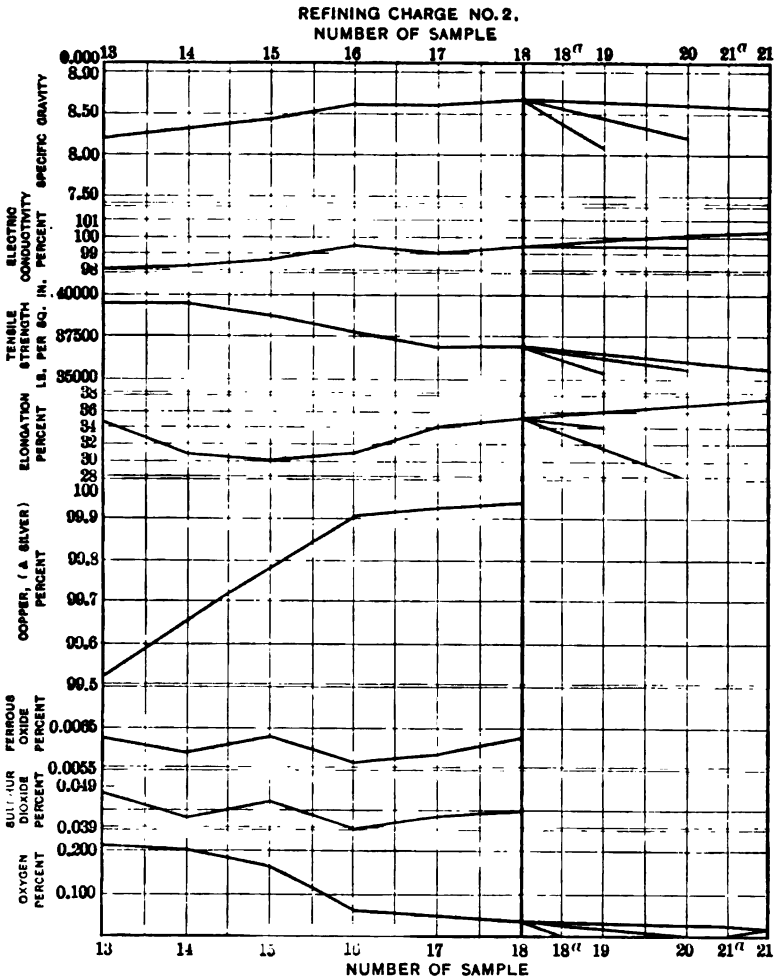

FIG. 21.—PHYSICAL AND CHEMICAL CHANGES OF ELECTROLYTIC COPPER IN REFINING.

the oxidizing fusion had not been carried as far as is common practice.

The specific gravity of the metal increases as the poling progresses, just as did the percentage of copper, more quickly during the first than during the second stage of poling; be-

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tween samples Nos. 13 and 16 there is a rise from 8.23 to 8.63, and between samples Nos. 16 and 18 a difference of only 0.06.

The curve of electric conductivity resembles that of copper-content and of specific gravity. Set copper, when annealed, had a conductivity of 98.2 per cent.; this increased at first quickly, reaching 99.6 per cent. with sample No. 16; tough-pitch copper showed no improvement upon this amount.

The tensile strength decreased very little considering the amount of oxygen that had been removed; at the start it was 39,130 lb., at the finish 37,020 lb.; the fall in tenacity is more gradual and regular than was expected.

The data for elongation are irregular. There is a fall from 34.6 to 30.1 per cent., then a rise of a similar amount to 34.2 per cent., followed by a slight increase to 35.1 per cent. It was expected that the elongation would increase with the elimination of oxygen.

3. *Summary of Refining-Charges Nos.* 1 *and* 2.

The two charges examined were electrolytic copper from the multiple process; they represented a high-grade metal and were refined in reverberatory furnaces of similar construction and capacity, and by the usual method of oxidizing with compressed air and reducing with the use of poles. It was therefore to be expected that the changes the metal underwent in poling would be similar. A comparison of the curves in Figs. 15 and 21 proves this to be the case. The percentage of copper rises quickly at first and at about the same rate as the cuprous oxide is reduced; later it increases more slowly as it becomes more difficult to deoxidize the remaining small amounts of cuprous oxide to just the quantity that has to remain with the tough-pitch copper. The amount of ferrous oxide present in electrolytic copper is very small; any excess over a minimum, varying with different charges, is quickly eliminated. The sulphur-content of electrolytic copper remains practically unchanged in fire-refining. The specific gravity and electric conductivity rise and fall with the copper-content; cuprous oxide has an effect opposite to that of copper. The tensile strength decreases as the reduction of cuprous oxide progresses; the corresponding increase in elongation is not shown by the curves as clearly as was expected.

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III. Studies in Overpoling Electrolytic Copper.

The current meaning of the term, copper overpoled in the reverberatory furnace, is that poling has been carried beyond the tough-pitch stage, with the result that the reduction has been carried too far, causing the copper to become porous and brittle, and thus unfit for industrial purposes. It will be shown that the brittleness of furnace-overpoled electrolytic copper must generally be attributed to other causes than over-reduction. The present investigation, dealing with such pure metal as electrolytic copper, excluded the consideration of the effects that elements like arsenic, antimony, lead, bismuth, nickel, etc., might have if present in the oxidized or the metallic state; it confined itself to the remaining active agents, cuprous oxide, gases and temperatures, and incidentally to sulphur and iron. The plan of work was to examine samples of tough-pitch and furnace-overpoled copper from the same charges as obtained from works, to eliminate all the oxygen from the tough-pitch copper by reduction in a crucible, and to compare the results.

(*a*) *Samples.*—In addition to the samples from the two refining-charges (Nos. VIII. and IX., Table IX.) discussed in the first part of this paper, there were examined five specimens of electrolytic copper from Eastern works (samples Nos. I., III., IV., V., VI.), one of casting-copper (sample No. VII.), and one of native copper (sample No. X.). The samples (Table IX.) are marked with Roman numerals; the letter A affixed to a numeral designates the sample as tough-pitch copper furnace-overpoled at the works, the letter B as tough-pitch copper crucible-overpoled in the experiments. The results are assembled in Tables IX and X. Further data in regard to furnace-overpoling are given in Table XI., in which have been brought together some facts of an experimental run made by a Western plant in 1899 with a charge of 27.5 tons of cathode copper. The charge was brought to the tough-pitch stage in the usual way, overpoled, rabbled again to convert the overpoled copper into set copper, poled to tough-pitch copper, and again overpoled.

(*b*) *Crucible-Overpoling.*—The object of overpoling in a crucible was to eliminate by means of charcoal and by the exclusion of air all the oxygen of a sample and thus obtain what may be

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termed true overpoled copper. The apparatus used is shown in Fig. 22. The reducing crucible, *A*, was made of Acheson graphite, which is practically free from impurities. The cavity, $\frac{1}{16}$ in. in diameter and 3.5 in. deep, was bored into a stick 1.5 in. in diameter and 4 in. long. The graphite crucible was placed in a size G fire-clay crucible, *B*, packed with crushed fire-brick, *C*, the tops of the graphite crucible and the packing were covered with a layer of charcoal, *D*, 0.5 to 0.75 in. deep, and the clay crucible closed with a lid. Filings and chippings from

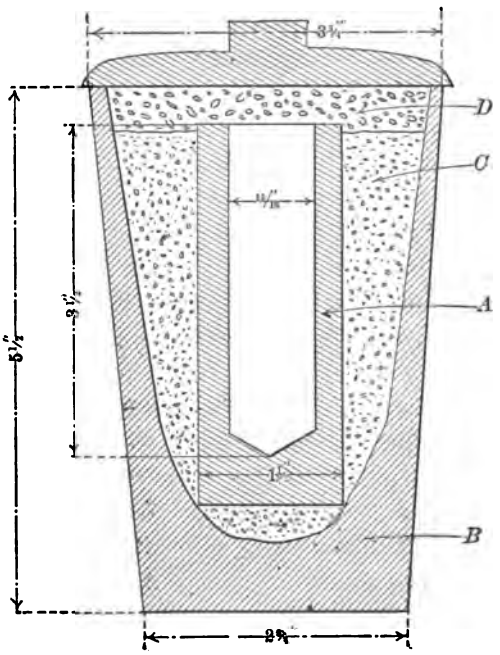

FIG. 22.—APPARATUS FOR CRUCIBLE-OVERPOLING.

tough-pitch copper were charged, and the apparatus then placed in a pot-furnace. When the charges had been fused, more copper was added to about fill the graphite crucible, the layer of charcoal spread over it, the copper kept molten about 15 min., the apparatus removed from the furnace, and allowed to cool slowly with a layer of charcoal still on top of the copper. When cold, the copper cylinder could be easily removed. A graphite crucible was found to stand four or five heatings without cracking. As air was not wholly excluded at first during the melting-down of the copper, the upper edge of the graphite crucible was

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TABLE IX.—*Record of Experiments.*

Kind of Copper.	No. of Sample	Physical Properties.							Chemical Composition.				
		Surface.	Resistance to Breaking	Appearance of Fracture.			Specific Gravity.	Electrical Conductivity.	Cu (+ Ag).	FeO.	SO_3.	Sulphide Sulphur.	O.
				Texture.	Luster.	Color.							
									Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.
Tough-pitch, electrolytic	I.	Even.	Very tough	Hackly.	Silky.	Rose.	8.666	99.6	99.819	0.004	0.017	.....	some.
Furnace-overpoled, electrolytic	IA.	Even, crowned.	Brittle.	Fine granular to vesicular.	Dull.	Light brick-red.	8.680	99.6	99.770	0.005	0.0038	present.	> I.
Crucible-overpoled, electrolytic	IB.	Flat, rough, cavity.	Tough.	Fibrous to hackly.	Half-silky.	Rose to light salmon.	9.043	98.92	.....	.....	.....	.....	none.
Tough-pitch, electrolytic	III.	Even.	Very tough	Hackly.	Silky.	Rose.	8.630	99.81	99.920	trace.	0.016	.....	some.
Crucible-overpoled, electrolytic	IIIB.	Crowned, cavity.	Tough.	Hackly to cubical.	Glimmering.	Light salmon to rose.	8.881	98.5	.....	.....	....	present.	none.
Tough-pitch, electrolytic	IV.	Even.	Very tough	Hackly.	Silky.	Rose.	8.586	100.57	99.942	0.029	0.020	.....	considerable.
Crucible-overpoled, electrolytic	IVB.	Crowned, cavity.	Very tough	Hackly.	Shining.	Rose to light salmon.	8.933	99.8	.....	.....	.....	absent.	none.
Tough-pitch, electrolytic	V.	Even.	Very tough	Hackly.	Silky.	Rose.	8.806	98.6	99.933	0.019	trace.	.....	some.
Crucible-overpoled, electrolytic	VB.	Crowned, cavity.	Very tough	Hackly to radiating.	Shining.	Rose to light salmon.	8.855	93.1	.....	.....	.....	absent.	none.
Tough-pitch, electrolytic	VI.	Even.	Very tough	Hackly.	Silky.	Rose.	8.550	100.0	99.889	trace.	0.013	.....	some.
Furnace-overpoled, electrolytic	VIA.	Smooth.	Very brittle	Finely granular.	Dull.	Light brick-red.	8.407	99.08	99.771	0.011	trace.	.....	> VI.
Crucible-overpoled, electrolytic	VIB.	Flat, rough, cavity.	Very tough	Hackly.	Shining.	Rose to light salmon	8.950	99.6	.....	.....	.....	absent.	none.
Tough-pitch, casting	VII.	Even.	Very tough	Hackly.	Silky.	Rose.	8.548	92.8	99.794	0.011	0.013	.....	some.
Crucible-overpoled, casting	VIIB.	Uneven, cavity.	Tough.	Hackly to slightly gran.	Dull.	Light salmon to rose.	8.930	93.5	.....	.....	.....	absent.	none.
Tough-pitch, electrolytic	VIII.	Even.	Very tough	Hackly.	Silky.	Rose.	8.547	99.6	99.952	0.023	0.013	.....	0.044
Furnace-overpoled, electrolytic	VIIIA.	Ridge.	Brittle.	Finely granular.	Dull.	Light brick-red.	8.240	99.8	.....	0.032	0.014	.....	0.0063
Crucible-overpoled, electrolytic	VIIIB.	Crowned.	Tough.	Hackly to columnar.	Glimmering.	Rose to light salmon.	9.086	99.3	.....	.....	.....	absent.	none.
Tough-pitch, electrolytic	IX.	Even.	Very tough	Hackly.	Silky.	Rose.	8.906	101.3	99.87	0.019	0.028	.....	0.0119
Furnace-overpoled, electrolytic	IXA.	Ridge.	Brittle.	Finely granular.	Dull.	Light brick-red.	8.824	100.7	99.70	0.058	0.021	.....	0.0794
Crucible-overpoled, electrolytic	IXB.	Crowned, cavity.	Tough.	Vesicular.	Bright.	Yellow.	.....	.....	.....	0.035	.....	present.	none.
Native copper	X.	.....	Very tough	Hackly.	Bright.	Rose.	8.830	102.5	99.960	0.0055 Fe.	none.	absent.	none.

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slightly burnt away. The entire absence of oxygen from the overpoled copper shows that the charcoal cover added had reduced any surface-oxidation of the charge that might have taken place. In melting down the first sample of tough-pitch copper, charcoal was charged with the copper. It was found, however, that some of the finer particles did not rise to the surface and made the copper cylinder rough and pitted. The main results are given in Table IX.; additional details of the tests are recorded in Table X.

TABLE X.—*Crucible-Overpoled Copper.*

Sample No.	Diameter.	Length.	Weight.	Surface.	Cavities, No. and Location.	Fern-like Crystals.	Vertical Section Through Copper Cylinder.
	Inch.	Inch.	Grams.				
IB	1 3/8	2 7/16	162	Flat, rough.	2 in top.	Indistinct.	
IIIB	1 1/8	2 5/8	165	Crowned.	1 in side.	Numerous.	
IVB	1 1/8	2 3/4	180	Crowned.	1 in top.	Few.	
VB	1 1/8	2 11/16	185	Crowned.	1 in side.	Numerous.	
VIB	1 1/8	2 5/8	152	Flat, rough.	1 in top.	Numerous.	
VIIB	1 1/8	2 1/2	170	Uneven.	1 in top.	Numerous.	
VIIIB	1 1/8	2 5/8	197	Crowned.	Not clearly defined.	Indistinct.	
IXB	1 1/8	2 3/4	190	Crowned.	1 in top.	Numerous.	

Table IX. shows that some samples of crucible-overpoled copper contain sulphide-sulphur. Its presence was determined by etching a polished sample with hydrofluoric acid. Under the microscope both cuprous oxide and cuprous sulphide show a bluish color, but, as first shown by Heyn,[12] etching with hydrofluoric acid colors cuprous oxide black and leaves cuprous sulphide unchanged, thus making it easy to distinguish them from one another. Fig. 29 represents crucible-overpoled copper free from oxygen, with black spots of cuprous sulphide. In the experiments, Baker and Adamson's c.p. hydrofluoric acid was used; 5 seconds' treatment was sufficient to change cuprous oxide from blue to black. Attention may be called to

[12] *Metallurgie*, vol. iii., p. 73 (1906).

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TABLE XI.—*Refining-Experiments in Reverberatory Furnace.*

Operations, December, 1899, on 27.5 Tons Cathode Copper.			Sample.		Physical Changes.				Chemical Changes.		
Character.	Time Given. Hr. and Min.	Temperature. °C.	Number.	Description.	Electric Conductivity, No. 12 B. & S. Gauge, Hard Drawn.	Tensile Strength. Lb. Per Sq. In.	Elongation in 5 Feet.	Torsion in 6 Inches.	As and Sb.	O.	S.
							Per Ct.				
Melting and skimming	8.50	.....	1	Skimmed copper.	could	not be	drawn.	.....	0.0068	0.352	0.0202
Rabbling	1.50	1,130	2	Set copper.	could	not be	drawn.	.....	.....	0.546	0.0078
Tough-poling	1.30	1,170	3	Tough-pitch copper.	96.57	65,400	1.1	80	0.0060	0.071	0.0066
Overpoling	1.05	1,190	4a	Overpoled copper, median line.	could	not be	drawn.	.....	.....	0.0609	0.0077
Overpoling	0.15	1,115	5b	Overpoled copper, spew.	96.00	65,000	1.1	32	.....	0.0540	0.0082
Rabbling	1.35	1,117 aver., 1,030 finish.	6c	Set copper.	could	not be	drawn.	.....	.....	0.481	0.0060
Tough-poling	1.25	1,120 aver., 1,155 finish.	7	Tough-pitch copper.	96.36	65,900	1.1	30	0.0070	0.058	0.0044
Dipping 86 bars, adding charcoal and brands	1.00	1,145	8	Tough-pitch copper.	96.13	66,600	1.2	27	.....	0.073	0.0044
Dipping 26 bars, adding charcoal and brands	0.28	1,120	9	Overpoled copper, crown.	96.10	66,500	1.2	28	.....	0.0506	0.0055
Overpoling	0.23	1,110	10	Overpoled copper, median ridge.	96.46	66,400	1.2	42	.....	0.0445	0.0087
Overpoling	0.05	1,130	11	Overpoled copper, spew.	96.48	66,700	1.1	42	0.0057	0.048	0.0103

(*a*) Between samples 4 and 5, charcoal raked off, air admitted, fresh coal added to fire.

(*b*) Between samples 5 and 7, charcoal raked off, fresh fire made on grate.

(*c*) Metal too cold, adheres to poles; poles removed, fresh fire made, and poling started again.

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the fact that Hampe[13] had ascertained long ago by analytical methods that cuprous oxide, cuprous sulphide and sulphur dioxide could be present together in tough-pitch copper.

The terms used in Table IX. to denote resistance to breaking require to be more closely specified. In breaking with a hammer a nicked sample clamped in a vise, the sample was termed very brittle when one blow was sufficient to break it, brittle when three or four blows were required, tough with more than four blows striking on one side, very tough with more than four blows striking on both sides.

The formation of a cavity when copper free from cuprous oxide is fused and cooled under reducing conditions, is a phenomenon to be expected, as copper shrinks upon cooling. Casting a bar of copper 3 by 3 by 30 in. on end with exclusion of air, J. B. Cooper[14] obtained a pipe 5 in. deep, as shown in Fig. 23. When air came in contact with the surface of such a bar, when partly solidified, the surface rose immediately and finally crowned. The cross-section of such a bar, Fig. 24, 7 in. beneath the top, showed a core about 1 in. in diameter, which was crystalline and porous, while the balance was solid, resembling native copper.

A similar experience is that of Percy[15]: Electrolytic copper melted under charcoal in a crucible and left to solidify therein showed no rise, but a depression. Similar copper melted under charcoal and poured, without taking any precautions to exclude air, gave a crowned surface.

While the samples used in the present experiments, weighing 162 to 197 g., did contain some cuprous oxide, and while its percentage may have been slightly increased during the first stage of fusion before the charcoal cover had been given, the microscopic examination of the specimens, which remained fused for 15 minutes under a charcoal cover and cooled in the crucible under it, showed that no cuprous oxide was present, and that therefore the reduction by charcoal had been complete. This is further brought out by the decrease in electric conductivity (see Table IX.) of all the high-grade samples of tough-pitch copper by crucible-overpoling, which must have re-

[13] *Op. cit.*, vol. xxi., p. 278 (1893).

[14] Private communication, September 3, 1905.

[15] *Metallurgy; Fuel, Fire-Clays, Copper, Zinc, Brass*, etc., pp. 275, 276 (1861).

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duced any oxide impurities present to the metallic state. A cavity was to be expected in the metal free from oxide, but the surfaces of all but one specimen (see Table X.) are also crowned. Crowning is due to the evolution of gases. That part of the gases held in solution by the tough-pitch copper have been completely eliminated in crucible-overpoling with the reduction of cuprous oxide and have not left the crucible-overpoled copper porous, is seen (Table IX.) by the rise in specific gravity of all the specimens from tough-pitch to crucible-overpoled copper. The samples of crucible-overpoled electrolytic copper then

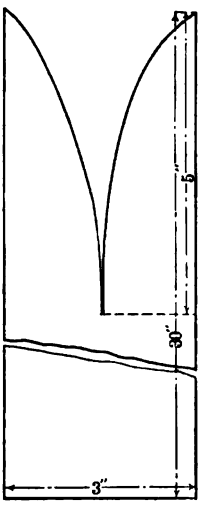

FIG. 23.—VERTICAL SECTION OF BAR CAST UPRIGHT UNDER REDUCING CONDITIONS.

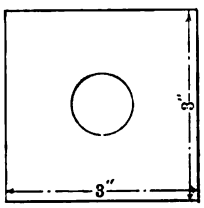

FIG. 24.—HORIZONTAL SECTION OF BAR, FIG. 23, AIR ADMITTED WHEN PARTLY SOLIDIFIED.

present the combination of a cavity due to cooling and a crown due to the evolution of gas.

Crucible-overpoled copper and native copper have only this in common, that they are both free from oxygen.

(*c*) *Furnace-Overpoling.*—When tough-pitch copper is cast from the reverberatory furnace into a mold, it gives an ingot, bar or cake with a level surface; when its surface shows a slight crowning, a ridge, or throws a worm (spews), it is sure to be overpoled. As shown above, the rising of the surface is due to the giving off of gas. Hampe[16] found that copper had the

[16] *Zeitschrift für Berg-, Hütten- und Salinen-Wesen in Preussen*, vol. xxi., p. 274 (1873).

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property of absorbing sulphur dioxide, hydrogen and carbon monoxide, which rendered the metal porous. By heating copper charged with soluble gas in a current of carbon dioxide, which is insoluble, he expelled the dissolved gas and obtained a dense metal with a correspondingly higher specific gravity. Thus[17] he raised the specific gravity of copper from Mansfeld, containing 0.075 per cent. of oxygen, from 8.525 to 8.906 by fusing in a current of carbon dioxide, the chemical composition of the metal remaining unaffected.

Caron[18] had proved before Hampe that fused copper had the property of absorbing hydrogen and carbon monoxide.

Stein[19] recovered from porous copper, by gently heating in vacuo, first hydrogen, then carbon monoxide. Heyn[20] found that copper heated in a current of hydrogen to 600° C., became brittle and showed a decrease in specific gravity. Stahl[21] examined three samples of slightly overpoled copper free from sulphur, taken from reverberatory-furnace charges just before casting. With a copper-content of 99.924, 99.893 and 99.899 per cent., the specific gravity was, 8.342, 8.466 and 8.266, while the specific gravity of the tough-pitch copper of the works averaged over 8.900.

Perhaps the most striking examples of gas-absorption by copper while being poled are those shown in Table XII., given by Stahl,[22] which show a decrease of oxygen with a decrease of specific gravity, while just the reverse would have taken place had it not been for the gas-absorption.

In poling, the charring of the wood sets free water-vapor, which stirs the copper, and carbon monoxide, hydrogen and hydrocarbons, which become more or less disseminated through it. As long as the copper is heavily charged with cuprous oxide, carbon monoxide and hydrogen cannot be retained by the copper, as they are oxidized to carbon dioxide and water-vapor, which are insoluble in copper; hydrocarbons are decom-

[17] *Op. cit.*, vol. xxii., p. 131 (1874.)

[18] *Comptes rendus*, vol. lxiii., p. 1129 (1866); *Dingler Polytechnisches Journal*, vol. clxxxiii., p. 384 (1867).

[19] *Berg- und Hüttenmännische Zeitung*, vol. xl., p. 235 (1881).

[20] *Zeitschrift des Vereines deutscher Ingenieure*, vol. xliv., p. 508 (1900).

[21] *Ueber Raffination, Analyse und Eigenschaften des Kupfers*, Fieke, Altenau i. Harz, p. 52 (1886).

[22] *Op. cit.*, p. 55.

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TABLE XII.—*Effect of Dissolved Gas upon Specific Gravity of Copper.*

Charge No.	State of Copper.	Cu.	O.	Specific Gravity.
		Per Cent.	Per Cent.	
1	Dense-poled (a).	.....	0.210	8.916
	Tough-poled, ½ hour.	.....	0.123	8.851
	Tough-poled, 1¼ hours.	99.882	0.086	8.713
2	Dense-poled (a).	.....	0.186	8.895
	Tough-poled, ½ hour.	.....	0.164	8.887
	Tough-poled, 1 hour.	99.860	0.078	8.684
3	Dense-poled (a).	.....	0.198	8.903
	Tough-poled, ¾ hour.	.....	0.102	8.704
	Tough-poled, 1¾ hours.	.....	0.051	8.405
	Reoxidized.	99.776	0.209	8.907

(a) Dense-poling, which is common in German practice, means poling under partly oxidizing conditions with the object of expelling dissolved sulphur dioxide; it precedes tough-poling.

posed, the hydrogen is first oxidized and then the carbon. As the percentage of cuprous oxide decreases in poling, the oxidation of the gases diminishes, the absorption of carbon monoxide and hydrogen increases, and the finely-divided carbon from the decomposed hydrocarbons rises unoxidized to the surface of the metal-bath. The absorbing power of copper for gas increases with the temperature and the purity of the copper. According to Stahl,[23] the gas-absorption becomes evident before the oxygen of the copper has been reduced to 0.07 per cent.; in one instance he noticed it when the copper still contained 0.160 per cent. of oxygen. Hampe[24] found that the presence of the usual small amounts of impurity in copper did not affect the solubility of hydrogen, that the carbon monoxide was less soluble than hydrogen, and that cuprous oxide had no influence on the solubility of sulphur dioxide. Stahl's experiments[25] proved that lead, arsenic and phosphorus in amounts larger than common in refined copper decreased its dissolving power for gas; thus the addition of about 0.25 per cent. of lead or 0.4 + per cent. of arsenic or 0.024 per cent. of phosphorus toughened porous copper sufficiently to permit its being hammered, rolled or drawn.

[23] *Op. cit.*, p. 47; also, *Berg- und Hüttenmännische Zeitung*, vol. xlviii., p. 323 (1889); vol. lx., p. 77 (1901).

[24] *Zeitschrift für Berg-, Hütten- und Salinenwesen in Preussen*, vol. xxi., p. 274 (1873).

[25] *Op. cit.*, p. 59.

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The next question to be considered is, how does furnace-overpoled electrolytic copper differ chemically from tough-pitch electrolytic copper.

As to the oxygen-content, the data in Table IX. show that furnace-overpoled copper may have a higher or a lower percentage of oxygen than the corresponding tough-pitch copper. Thus, examples IA (Fig. 26), VIA and IXA (Fig. 28) contain more, and sample VIIIA less oxygen than the corresponding samples of tough-pitch copper I. (Fig. 25), VI., IX. (Fig. 27) and VIII. In Fig. 21, representing graphically the changes of sample No. VIII. (Table IX.), the tough-pitch copper, No. 18, was furnace-overpoled and three bars were cast at short intervals, giving specimens marked Nos. 19, 20 and 21, all of which contain less oxygen than No. 18. Crucible-overpoling Nos. 18 and 21 eliminated all the oxygen, as seen by Nos. 18a and 21a. In Table XI., the two furnace-overpoled samples contain less oxygen than the corresponding tough-pitch copper. Thus, of the six samples of furnace-overpoled copper, three contain more and three less oxygen than the respective tough-pitch copper. Refiners hold that most furnace-overpoled copper represents a false-overpole—*i. e.*, the oxygen-content has been raised. Nevertheless, it is not an uncommon practice to slightly rabble copper that shows signs of crowning or tendencies to spewing in order to correct the evil. The rabbling increases the oxygen-content of the copper and thus diminishes its dissolving power for gas This practice is in line with the facts recorded above, in regard to the solubility of gas in copper.

As to the sulphur-content, Table IX. gives three samples of furnace-overpoled copper with a lower (IA, VIA, IXA) and one with a higher (VIIIA) percentage of sulphur than the corresponding tough-pitch copper (I, VI, IX, VIII). In Table XI., the four analyses of furnace-overpoled copper give more sulphur than the two of tough-pitch copper.

In regard to iron, Table IX. shows that the four samples of furnace-overpoled copper contain more iron than the tough-pitch copper. Why this should be so is not clear.

The influence of temperature is not definitely settled by the evidence of Table XI. In the first test, the temperature of the metal bath rose during the 1 hr. 5 min. of overpoling from 1,170 to 1,190° C., the copper became overcharged with gases

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and showed a median line when cast in a bar; air was then admitted (which did not increase the percentage of cuprous oxide) and coal added to the fire, when after 15 minutes' additional poling at a temperature of 1,115° C., a bar cast spewed upon solidifying. These facts seem to prove that a temperature of 1,115° C. is sufficiently high for copper overpoled for 1 hr. 20 min. to hold enough excess-gas to make it spew upon solidifying when cast in the form of a bar.

In the second test, the copper, 1 hr. 51 min. after the tough-pitch stage, showed a ridge and then required only 5 minutes' poling at a temperature 20° C. higher than before to throw a worm.

The two periods of overpoling, 1 hr. 20 min. and 1 hr. 50 min., of course, are excessively long, as under normal conditions from 5 to 10 minutes is sufficient to spoil the pitch.

IV. Conclusion.

The evidence obtained as to oxygen-, sulphur-, and iron-content of furnace-overpoled electrolytic copper, and as to effect of temperature, does not point clearly in a single direction and permits various interpretations. There remain, however, as undisputed facts, that copper absorbs hydrogen, carbon monoxide and sulphur dioxide, and that the solubility increases with the temperature and decreases with the oxygen-content. With set copper the solubility of the gas is at a minimum on account of the low temperature and the high percentage of the oxygen of the metal bath, and set copper solidifies with depressed surface. With crucible-overpoled copper the solubility is at a maximum on account of the necessarily high temperature and the entire absence of oxygen.

Between these two extremes lies the level set or proper pitch of tough-pitch copper. The proper pitch then appears to be the resultant of the hollow pitch of set copper and the crowned pitch of overpoled copper, and to vary, independently of small admixtures of sulphur and iron, with the size of the casting; a heavy cake holding more gas requires a copper richer in oxygen to counteract the raising power of the gas than does a wire-bar.

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Grinding in Tube-Mills at the Waihi Gold-Mine, Waihi, New Zealand.

BY E. G. BANKS, WAIHI, AUCKLAND, NEW ZEALAND.

(New York Meeting, April, 1907.)

THIS paper is presented in the belief that metallurgists and chemists will be interested in the practice of grinding in tube-mills in connection with stamps, especially since the records of working here given extend over a lengthy period of time (since May, 1905).

The ore from the Waihi mine—more especially that produced in the upper levels—contains a large proportion of hard, chalcedonic quartz, and the gold exists in an exceedingly fine state, conditions which necessitate very fine crushing in order to obtain a high extraction of the precious metal.

Before the introduction of tube-mills at the 90-stamp Waihi mill, it was found necessary to stamp through 40-mesh (1,600 holes per sq. in.) woven wire-screens, having a fairly high discharge. The pulp then graded:

	Per Cent.		Per Cent.
On 50-mesh,	0.1	On 120-mesh,	8.77
On 60-mesh,	8.74	On 150-mesh,	7.48
On 80-mesh,	16.06	Passed 150-mesh,	55.19
On 100-mesh,	3.66		

The stamp-duty was 2.89 short tons per stamp per day, or a total of 260 tons daily.

Although the extraction on this pulp was: gold, from 88 to 90 per cent., and silver, from 74 to 78 per cent., it was recognized that finer grinding of the sands would prove beneficial, provided a machine could be found to do this work economically. Various grinding-mills and pans were tried, but without satisfactory results, the particles of sand being so hard that the capacity of any of the machines was too small to be economical.

The results of grinding in tube-mills in other countries were so satisfactory that three tube-mills were erected at the Waihi

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mill, the installation being completed in May, 1905, since which time an average duty of about 2.7 mills has been maintained. The mills are of the Davidsen, 22-ft. type, and are run at a speed of 27.5 rev. per min. Each mill is loaded with 5.5 tons of flints, and requires 50 h.p. to operate it. The mills are stopped for inspection and addition of flints once a week. The quantity of flints consumed is 18 cwt. per mill per week. In order to reduce the time required to charge the flints into the tube-mill, a new door is being fitted which will admit of two or three charges a week instead of only one, as formerly. In this way the weight of the flints in the mill can be kept at all times much nearer the weight of the original charge, 5.5 tons.

Various liners have been used, including "Silex" and "Delarue" quartzite blocks, and also cast-iron liners, 1.25 in. thick. The iron liners last about as long as the quartzite blocks—viz., 2.5 months—but the grinding-result is not so good. A new lining, invented and patented by Mr. H. P. Barry, called the "Honeycomb lining," is now being tried with very promising results. This liner consists of a light cast-iron frame, 22 by 14 by 3 in. deep, shaped to the curve of the mill. Thin walls divide this lining into four or six compartments. A temporary sheet-iron back is fastened to the frame, and each compartment is then firmly packed with rough lumps of hard quartz or quartzite, varying in size up to 4 in. square, bedded-in with a mixture of Portland cement, coarse sand and fine sand. The liners so formed are allowed to set, preferably under exhaust steam, for several weeks—the longer the better—before being placed in the mill. This method of lining calls for a much shorter stoppage than with the quartzite blocks. The frames fit neatly with each other and with the shell of the mill, and only a small quantity of cementing material is required.

If made with hard material these liners stand very well, and cost, including labor, about $175, as compared with $400 for lining with quartzite blocks. The grinding-efficiency of a mill with this new liner appears to be quite equal to that of one lined with quartzite blocks.

The stamps weigh 1,000 lb., and crush through 20-mesh screens. The proportion of water to ore is 10 to 1, and the output is 354 tons per day, which is equivalent to about 4 short tons per stamp.

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The pulp is lifted by wheel-elevators to four sizing-boxes, each 4 ft. square and 4 ft. deep; no upward flow is used. The slime and fine sand overflow and pass to the treatment-plant. The coarse sand, having 2 parts of water to 1 of sand, is divided into three portions and flows directly to the mills. It is intended to put in a de-watering-box at the head of each mill, with a view to improve the grinding.

The grade of the pulp, before and after the mill-treatment, is:

Size.	Before Grinding (90 Stamps, on 20-Mesh).		After Grinding in Three Tube-Mills.	
	Per Cent.	Tons.	Per Cent.	Tons.
On 30-mesh	5.32	18.85	0.03	0.11
On 40-mesh	9.77	34.56	0.12	0.40
On 60-mesh	15.94	56.42	1.13	4.01
On 100-mesh	13.96	49.42	7.43	26.28
On 150-mesh	12.29	43.50	18.42	65.22
Through 150-mesh	42.72	151.25	72.87	257.98

The daily tonnage of sands passing through the tube-mills is about 230 tons, or about 77 tons per mill.

It will be seen from the above grading that the mills are doing very good work, practically all the material of 30,- 40- and 60-mesh size having disappeared.

An additional tube-mill is being installed, and when completed either the coarser portion—up to 100 mesh—will be sent to this mill, or coarser screens, say 15-mesh, will be used on the stamps. It is a matter of experiment to determine which will give the better commercial result—finer grinding for increased extraction, or larger milling-tonnage. In addition to the benefit of increased tonnage by the substitution of 20-mesh screens in place of 40-mesh, the tube-mills have favorably influenced the extraction, for the reason that before their use the combined sand and slime residues assayed 31 grains of gold per ton, representing an extraction of 89.8 per cent., but after installation the combined residues assayed 24 grains of gold per ton, representing an extraction of over 92 per cent.

A most important result of grinding in tube-mills has been the effect on the slimes. A large proportion of sand is ground so fine that it passes the *spitzlutte* with the slime, the result

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being that the slime is more easily treated either by the filter-presses or the vacuum-process. This result is shown by the time required for filling and washing the presses, which can now deal with 30 per cent. more slime than in treating slimes from stamps on 40-mesh size.

The cost of running the tube-mills, per ton of sand passed through the mills, is:

	Cents.
Power,	12.5
Flints and liners,	14.0
Labor, repairs and stores,	1.5
Total,	28.0

or, on the total mill-tonnage, 18.2c. per ton of ore crushed.

The chief benefits derived from tube-mills at Waihi are:

1. Increased extraction, amounting to about 36c. per ton on the whole of the ore crushed.

2. Increased tonnage of fully 36 per cent.

3. A saving of 75 per cent. on the cost of screens. The 20-mesh now used costs less and lasts considerably longer than the 40-mesh previously used.

4. Amalgamation improved by from 5 to 7 per cent.

5. The slime, owing to the contained fine sand, is more easily treated.

The reduction in milling-cost due to the tube-mills is fully 12c. per ton on the total tonnage. This, together with the 36c. improved extraction, represents a total increased saving of 48c. per ton, or $169 per day on the 90 stamps.

When it is considered that, in addition to this result, the bullion production is augmented by the product from the extra 94 tons per day, it must be conceded that tube-mills have proved highly successful at Waihi.

Owing to these results at the Waihi mill, arrangements are now in hand to equip the Victoria mill of 200 stamps with a plant of at least nine tube-mills.

By the improved methods now coming to the front for handling slimes, and the economical grinding which is obtained with tube-mills, it is my opinion that the time is not far distant when such ores as those at the Waihi mine will be treated mostly in the form of slime.

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The Butters Slime-Filter at the Cyanide Plant of the Combination Mines Company, Goldfield, Nev.

BY MARK R. LAMB, GOLDFIELD, NEV.

(New York Meeting, April, 1907.)

THE treatment of slime is of special interest to those engaged in cyaniding gold- and silver-ores. The usual practice is to make as small a percentage of slime as possible. In many instances the slime is given no treatment, but is impounded in dams in the hope that the future will develop some method of economically treating this product. The filter-press was the first step upward from ordinary decantation, but, on account of heavy labor-charge and high cost, its use has been limited to high-grade material.

The slime at the Combination mill at Goldfield averages perhaps $20 per ton, and although the values are quickly dissolved the filter-press installation was not entirely satisfactory, resulting in the erection of the canvas-cell filter, developed by Chas. Butters and his staff. This filter is a great improvement over the filter-press, and the following description of it will be of interest to those engaged in treating slime produced in crushing ores, especially in view of the fact that, by the use of this filter, slime can be treated at a lesser cost and with a higher percentage of gold- or silver-extraction than in the ordinary treatment of sand; and, moreover, the initial outlay for an all-sliming plant is less than that for the ordinary sand- and slime-plant.

The economy of construction of the Butters filter-plant is clearly shown in Figs. 1 and 2, which illustrate the 20-frame filter of 40 tons capacity, now being installed by the Nevada Goldfield Reduction Co. In this construction the slime-pump is so connected that it can pump to or from either tank or the filter by changing the valve-settings—a combination which is necessary in this particular installation, since sufficient fall is not available for filling and discharging the filter by gravity.

Ordinarily, a slime-plant comprises: a filter-box with frames

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or cells; two tanks of double the capacity of the filter-box, for slime and water, respectively; a "wet" or "dry" vacuum-pump, or other source of vacuum; a centrifugal slime-pump; and a series of agitation-tanks, which also provides storage for

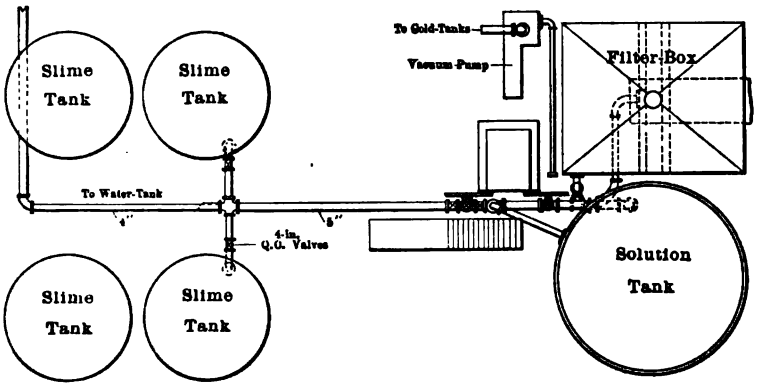

FIG. 1.—THE 20-FRAME FILTER OF THE NEVADA GOLDFIELD REDUCTION CO. (Plan.)

the slime. With regard to the economy in labor required to operate the Butters filter-plant, 500 tons of slimes, or even more, can be treated, filtered and discharged by one man per

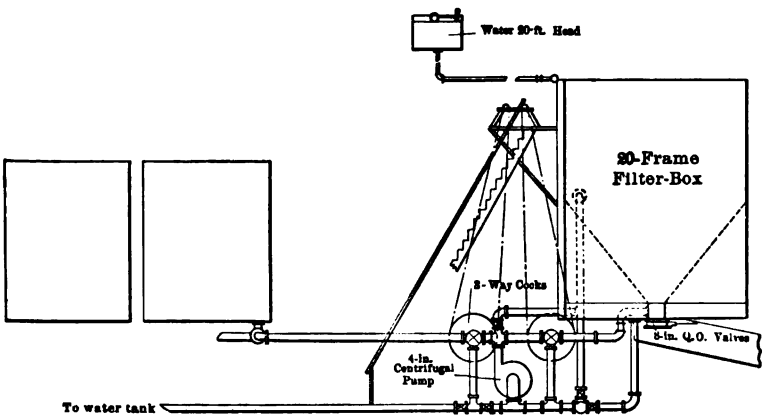

FIG. 2.—THE 20-FRAME FILTER OF THE NEVADA GOLDFIELD REDUCTION CO. (Side Elevation.)

shift—a remarkable gain as compared with ordinary practice in filter-pressing.

Where the filter is installed to replace treatment by decantation, no extra tanks are required, and the necessary additions are merely the filter and a source of vacuum. Furthermore,

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the unused settling- and treatment-tanks can be used as treatment-tanks, thus increasing the time of treatment, if desirable, and also increasing the capacity of the plant.

The cycle of operations is as follows: The slime, after agitation in solution the required time, is pumped into the filter-box. As soon as the latter is full the filter is connected to the source of vacuum and the gold-solution drawn from the pulp through the canvas of the cells, while the slime forms a layer on the outer surface of the canvas. While the slime-cake is being formed, the filter-box is kept full of slime-pulp by pumping it in as fast as solution is drawn out through the frames. When this layer is of suitable thickness (which depends on the permeability of the slime) the vacuum is reduced to a pressure barely sufficient to hold the slime in place, and the pulp still in the filter-box is returned to its storage-tank. The centrifugal pump is set to fill the filter-box with water, the vacuum is raised, and the slime-cake is washed. This water which passes through the filter is sent to the gold-tank. When desirable, the slime can be given a wash with solution before washing with water. This will be done at the Nevada-Goldfield plant. When all dissolved metal is removed from the cake it is dropped from the canvas by merely breaking the vacuum and turning water or water and air into the cells under a pressure of about 10 lb. per sq. in. A period of 5 min. suffices to drop the slime. Surplus water in the filter is then drawn off, if the saving of small quantities of water is desirable, and the bottom discharge-valve opened. The slime, containing from 20 to 40 per cent. of moisture, is discharged in less than a minute.

The time required for the treatment of one lot of slime is about 3 hr., depending upon the thickness of the cake, the size of the slime-pump, and the permeability of the material—all matters which can be determined in advance. The filter is used six or eight times in 24 hr. Correctly speaking, the slime is not treated in the filter, since agitation and extraction of the values take place before the filter is reached. In other words, the filter is used solely to displace the metal-bearing solution. In Mexico, the filter is handled by one Mexican peon per shift. These peons learn quickly and are entirely satisfactory.

One of the most important and vital features of the filter is the fact that the regulation of the thickness of the layer of

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slime depends upon its permeability. Thus, if fine sand is mixed with slime, it tends to collect on the bottom of the center cells (which are directly over the inlet from the pump), but in a proportionately thicker layer, thus causing all parts of the layer of slime to be equally washed.

Compare this ideal automatism with the action in an ordinary filter-press, in which the slime and sand have a decided tendency to classify in the frames, with the result that the cake will show a larger proportion of sand in the lower half. This settling, of course, makes it necessary to wash the charge longer than would be the case if the cakes were homogeneous throughout. In many instances the entire volume of wash-solution passes through the coarser material in the lower portion of the frame of the filter-press, and the fine slime is not washed. This results in a large volume of wash-solution, besides loss of cyanide and dissolved metals.

The economy in time for complete displacement by the Butters filter (from 15 to 20 minutes, see Table I.) is plainly evident when compared with the usual practice, using a press or by decantation. The Virginia City plant of Chas. Butters & Co. is now treating ores from Tonopah, the slime of which settles very slowly. The pulp is brought to the plant with about 100 parts of water to one of slime, which condition will be changed shortly, but which now makes a secondary de-watering filter necessary. This latter device works well, and but for it the slime could not be treated except with a large loss of solution.

According to the paper by Charles Butters and E. M. Hamilton, entitled "On the Cyaniding of Ore at El Oro, Mexico, Dealing Principally with Re-Grinding of Sands,"[1] describing practice at El Oro, Mexico, slime can be made of sand at a cost of $0.53 gold per ton. The cost at the Combination mill is a little less than this figure.

It is rarely the case that the difference in value between sand and slime tail-assays would not exceed this amount, and including the occasional (?) slimy sand-tank, with which the cyanider must contend, calculation will show that many plants could largely increase profits by sliming the entire product.

Fig. 3 is a view in the Virginia City plant, showing the elec-

[1] *Transactions of the Institution of Mining and Metallurgy*, vol. xiv., pp. 3 to 46 (1904-05).

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trolytic precipitation-vats in the foreground, directly in front of a 90-frame filter-box, which has a capacity of from 150 to 200 tons per day.

Fig. 4 shows the slime-pump, three-way cocks, quick-opening valves and piping at the Combination plant, which arrangement, as previously explained, is necessary only where sufficient grade is not available.

Fig. 5 shows the end-view of the lower half of the filter-box, and, at the right-hand side, the geared vacuum-pump.

Fig. 6 shows one leaf of the filter, which has lines of stitching in order to resist the internal pressure.

Fig. 7 is a perspective view of a plant of 100 tons daily capacity, comprising the filter-box, the slime- and water-storage tanks above, the slime-storage tank below, the vacuum-drum connected to the filter-leaves and to the gold-sump, and the gold-solution and slime-pumps.

Fig. 8 is a detail view of the filter-box shown in Fig. 7, with filter-leaves in position, some shown in section, and some removed.

The list of tests, made to show results which were being attained at the Combination plant,[2] is given below.

It should be explained that "water" wash was, in reality, solution assaying $0.60 per ton, which would otherwise be waste solution. This water, as it is drawn through the slime-cake, does not go to the zinc-boxes, but is used to replace the weak wash, which is precipitated. The weak wash assayed $4.02 per ton.

TABLE I.—*Solutions from a Cake* 1.25 *In. Thick.*

Weak Wash.		*Water Wash.*	
Time.	Value Per Ton.	Time.	Value Per Ton.
After 2 min.,	$12.22	After 5 min.,	$3.66
After 4 min.,	12.20	After 10 min.,	3.62
After 6 min.,	11.89	After 15 min.,	3.12
After 8 min.,	11.96	After 20 min.,	0.60
After 10 min.,	11.40	After 25 min.,	0.60
After 12 min,	10.90	After 30 min.,	0.60
After 14 min.,	8.90		
After 16 min.,	6.82		
After 18 min.,	4.76		
After 20 min.,	4.46		
After 22 min.,	4.26		
After 24 min.,	4.12		
After 26 min.,	4.10		
After 28 min.,	4.02		

[2] *Engineering and Mining Journal*, vol. lxxxi., pp. 1236 to 1238 (1906).

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FIG. 3.—THE 90-FRAME FILTER-BOX AND ELECTROLYTIC PRECIPITATION-VATS AT VIRGINIA CITY PLANT.

FIG. 4.—SLIME-PUMP, THREE-WAY COCKS, VALVES AND PIPING AT COMBINATION PLANT.

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FIG. 5.—END-VIEW OF LOWER HALF OF FILTER-BOX, WITH GEARED WET VACUUM PUMP AT THE SIDE, COMBINATION PLANT.

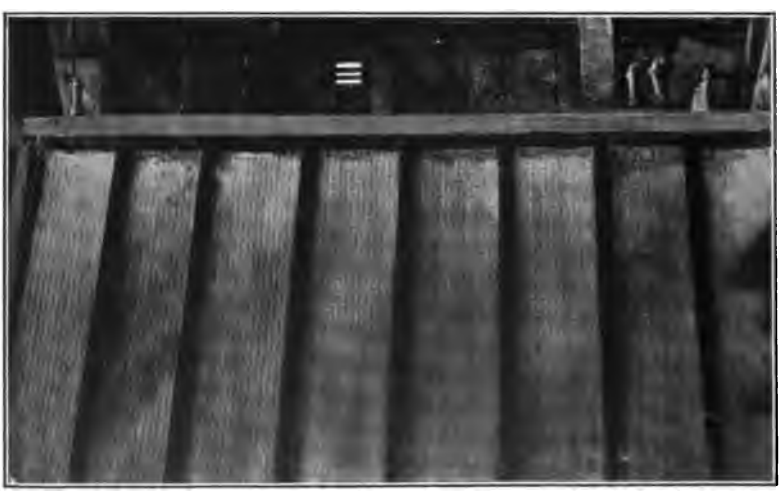

FIG. 6.—ONE LEAF OF THE FILTER, SHOWING REINFORCEMENT BY STITCHING, COMBINATION PLANT.

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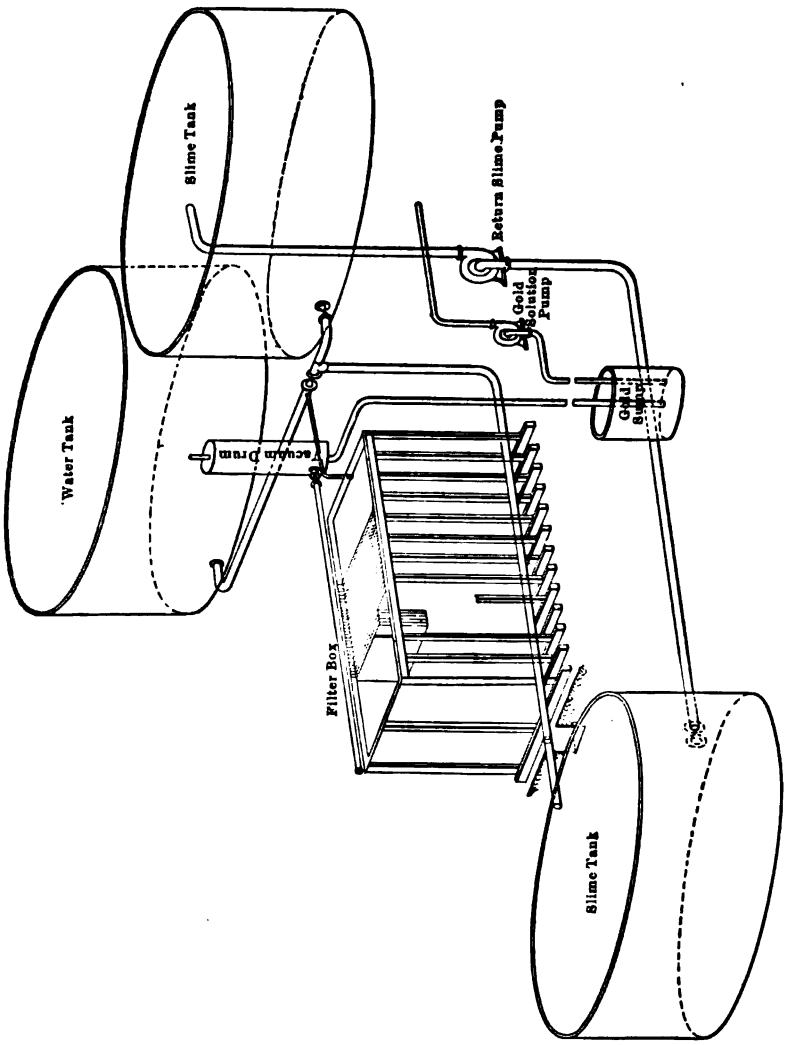

FIG. 7.—VIEW SHOWING ARRANGEMENT OF A 100-TON SLIME-FILTER.

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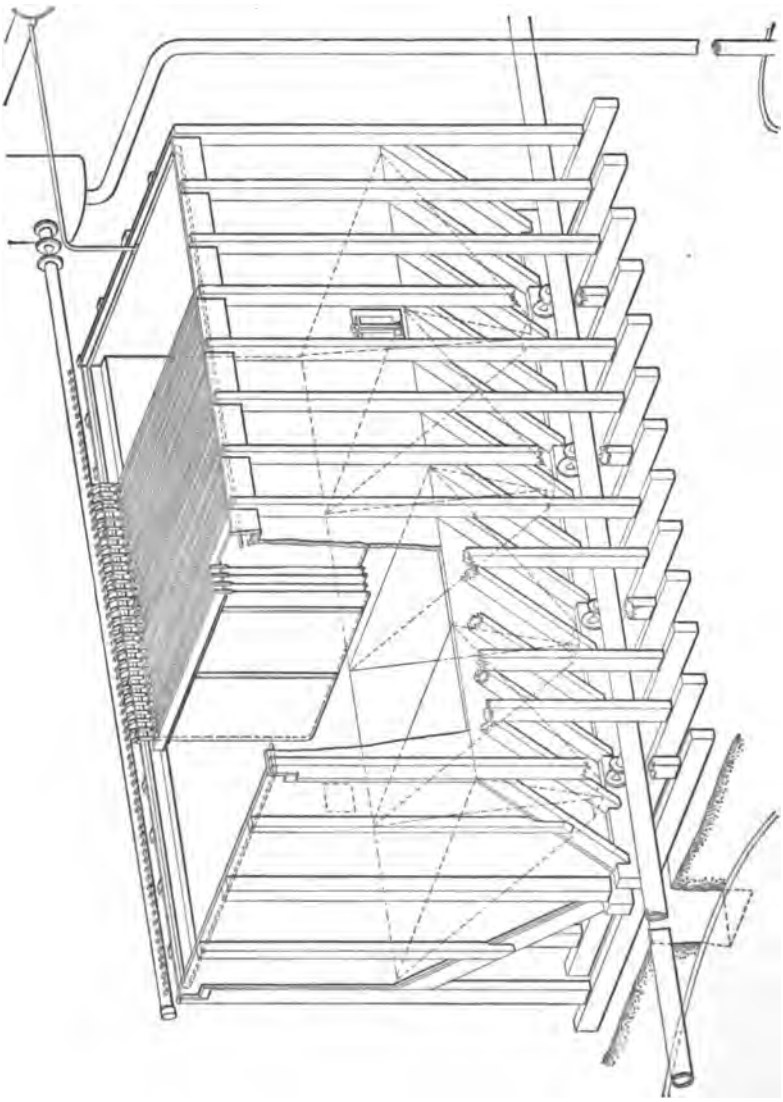

FIG. 8.—DETAIL VIEW OF FILTER-BOX.

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The practice at the Combination mill is to wash from 16 to 18 min. with solution and the same length of time with water.

The ordinary slime-settling device, whether tank or *spitzkasten*, rarely produces a product with less than 60 per cent. of moisture, and in many cases the proportion is three or four to one. This water must necessarily dilute the cyanide solution added to the slime, or, in case cyanide is added to the slime without further addition of solution, this solution, after precipitation, must be run to waste. Of course, the ideal way is to mill in solution, though for other reasons this is not always desirable.

When milling in water, by a proper arrangement of pipes and valves, and by providing a filter of sufficient capacity, the settled slime can be dried to any economical degree of moisture; and by filling the filter-box with solution the slime can be pulped again by means of the centrifugal pump and returned to the agitators, thus avoiding the necessity of making solution which must be run to waste.

As an example of what can be done, 100 tons of slime, it treated without removing the water, will cause the loss of about 75 tons of weak solution, containing about 0.8 lb. of KCN per ton, or 60 lb. If the Butters filter is used, this loss can easily be reduced to 8 lb. or less, without additional expense.

Particular attention should be paid to the thoroughness of the wash. After 20 min., if pure water be used, the wash-water from the slime at the Combination mill shows no measurable quantity of cyanide.

The quantity of solution made by washing is much reduced in volume when compared with the amount necessary for washing a filter-press charge. In fact, the wash-water will about make up for evaporation losses when ore is milled in solution.

Experiments are now under way with a view to doing the entire treatment on the filter, thus saving tankage and time. These experiments will only succeed with ores such as that treated at the Homestake slime-plant, where extraction from slime is almost instantaneous. Such treatment would not succeed with a silver-ore, for example, where long agitation and aëration are required.

I feel confident that this filter will be the cause of many plants abandoning entirely the treatment of sand, thus reducing the cost of installation, depreciation and operation, and increasing the capacity and the extraction.

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Velocity of Galena and Quartz Falling in Water.

BY ROBERT H. RICHARDS, MASSACHUSETTS INSTITUTE OF TECHNOLOGY, BOSTON, MASS.

(New York Meeting, April, 1907.)

I. Introduction.

The object of this paper is to enlarge the field of settling velocities treated by me in my former papers, Close Sizing Before Jigging, and Sorting Before Sizing.[1] There seemed need of work both on coarser and on finer sizes.

Messrs. A. Sidney Warren and M. L. Nagel undertook the investigation for the coarser sizes, from 12.85 mm. down to 2.05 mm. diameter. Their work, because of the closer spacing and because of the increased number of observations and consequent stronger averages, called for a revision of the former work.

Messrs. G. A. Barnaby and Ralph Hayden undertook this part of the work, from 2.49 mm. down to 0.28 mm., which is near the limit of sifting. Their work covers the ground much more minutely and comprehensively than the former papers.

There remained to be covered the portion of the field too fine for investigation with the sieves. This part, covering the range of grains from 0.48 to 0.03 mm., was undertaken by Mr. E. S. Bardwell. For it he used the elutriation method.

II. The First, or Coarsest, Section of the Field.

The first desideratum in this investigation was a series of sieves for sizing the sands preparatory to making the tests. It became necessary, therefore, to decide on a sieve-scale for this purpose.

A sieve-scale is a series of sieves in which the dimensions of the holes of the successive members form a series increasing by a definite ratio. There are three natural sieve-scales: 1, progressing by doubling the width of the hole for each suc-

[1] *Trans.*, xxiv., 409 to 486 (1894), and xxvii., 76 to 106 (1897).

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cessive member of the series; 2, progressing by doubling the area of the hole for each successive member of the series, for which the multiplier is $\sqrt{2}$, or 1.41421; 3, progressing by doubling the volume of the grain for each successive member of the series. The multiplier for this is $\sqrt[3]{2}$, or 1.25993. The second scale is the one adopted by Rittinger. In the present work it was thought wise to diminish the intervals more than would ever be required by mill-work in order to meet the most exact demands for the settling-values of grains.

The sieve-scale chosen was obtained from the Rittinger scale by interpolating another sieve between each member of his series, and is called in the paper double Rittinger. The multiplier which produces this result is $\sqrt[4]{2}$, or 1.18921. (See Table I.)

TABLE I.—*The Three Natural Sieve-Scales and Double Rittinger.*

Double Width.	Double Area. Rittinger.	Double Volume.	Double Rittinger.
1.	1.	1.	1.
			1.18921
		1.25993	
	1.41421		1.41422
		1.58742	
			1.68180
2.	2.	2.	2.
			2.37842
		2.51986	
	2.82842		2.82844
		3.17484	
			3.36360
4.	4.	4.	4.
			4.75684
		5.03972	
	5.65684		5.65688
		6.34969	
			6.72721
8.	8.	8.	8.
			9.51368
		10.07944	
	11.31368		11.31376
		12.69938	
			13.45443
16.	16.	16.	16.

In order to produce this double Rittinger sieve-scale, the values given in Tyler's catalogue of screens were studied and

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the sieves that were nearest to double Rittinger values were taken. Table II. gives, in the first column, double Rittinger figures; in the second column, the nearest values from Tyler's catalogue. The third and fourth columns are Warren's and Nagel's actual measurements of the holes in the Tyler screens. The fifth column is the average of Nagel's and Warren's figures, which is adopted as defining the sizes of grains used in the tests. The sixth is the average of each screen size with the one above it, and therefore gives approximately the average of the diameters of the grains which would pass through the coarser screen and rest on the finer one in each case.

TABLE II.—*Double Rittinger Sieve-Scale Used by Warren and Nagel.*

Double Rittinger.	Tyler's Catalogue.	Nagel's Measurement.	Warren's Measurement.	Average of Nagel and Warren.	Average of Screen with One Above.
		Width of Holes in Millimeters.			
13.454	.....	12.80	12.90	12.850	.....
11.314	11.210	11.01	11.01	11.010	11.930
9.514	9.696	9.49	9.52	9.505	10.257
8.000	8.047	7.80	7.78	7.790	8.647
6.727	6.751	6.83	6.86	6.845	7.317
5.657	5.786	6.00	5.94	5.970	6.407
4.757	4.690	4.94	4.95	4.945	5.457
4.000	3.923	4.21	4.21	4.210	4.577
3.363	3.299	4.15	4.10	4.125	4.167
2.828	2.818	2.84	2.81	2.825	3.475
2.378	2.360	2.83	2.81	2.820	2.823
2.000	1.980	2.07	2.04	2.055	2.437

In measuring the sieves Warren and Nagel found that the spaces were by no means all of exactly equal size, and were often rectangles instead of squares. First the diameters of the wires were measured with calipers. Six measures each by Warren and Nagel were obtained and the mean of the 12 measures was adopted. Next, a certain number of meshes was counted off on each screen and the distance across those meshes was measured in millimeters and tenths. From this measurement was subtracted the diameter of the screen-wire multiplied by the number of meshes, and this remainder was divided by the number of meshes. From six such measurements on each screen by each observer, an average was obtained giving the figures in columns three and four.

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Galena and quartz were selected for this investigation, partly because they are among the commonest minerals in the dressing-works, and also because very pure samples of each were available.

The galena used for this work was from the jigs of Joplin, Mo., and it was crushed by rolls to pass a screen with holes 13.450 mm. diameter. Determinations of its specific gravity were 7.49, 7.50, 7.56, 7.46; average, 7.50.

The quartz used for this work was plain white quartz purchased in the market. Its locality was not known. It was crushed to pass the above screen, and its specific gravity was tried four times, each giving 2.65 as the specific gravity.

The samples were next sized by screening them on the series of sieves, beginning with the coarsest and ending with the finest. Each size included grains which passed through the next coarser screen and failed to go through the next finer.

Experiments in testing the velocity of falling grains were made in a vertical tube, 11.5 ft. high and 8 in. in diameter, filled with water. This tube, Fig. 1, consisted of three sections: the upper one a flanged glass tube, *R*, 2 ft. high; the middle one a galvanized iron tube, *S*, flanged at each end, 7.5 ft. high; and the lower one a flanged glass jar, *T*, 2 ft. high.

The joint between the upper glass tube and the galvanized iron tube consisted of the glass flange, *C*, and the iron flange, *E*, an iron ring, *A*, a felt washer, *B*, a rubber gasket, *D*, and four bolts, *F*.

The joint between the lower end of the galvanized iron pipe and the flanged glass jar consisted of two flanges, *I* and *J*, the rubber gasket, *L*, a felt cushion, *G*, and a wooden plate, *H*, with four bolts, *K*, that extend the length of the jar.

The longest even distance that could be laid off conveniently for a measured course was 2800 mm.; 18 in. from the top of the tube two pasters, *N*, were put on the glass at front and back, so that the top edges give the upper sight and form a range for the starting-point. The space at the top was left to allow the grain to acquire full velocity; 2800 mm. below these pasters, two other pasters, *P*, were put on the lower jar to mark the finish. The top edges of these formed the lower sight. The top of the tube had a loosely-fitted zinc cover with a hole about 0.5 in. in diameter in the exact center. Through

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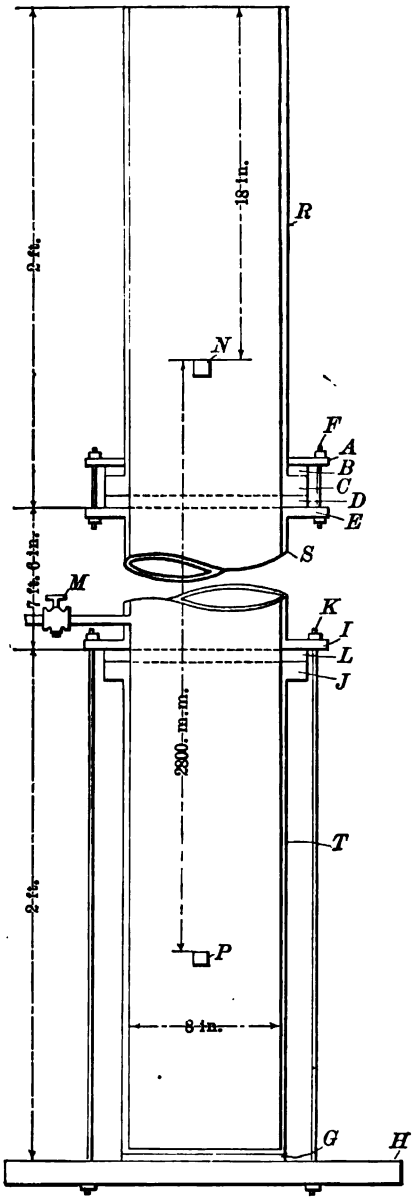

FIG. 1.—SORTING-TUBE FOR THE COARSER GROUP.

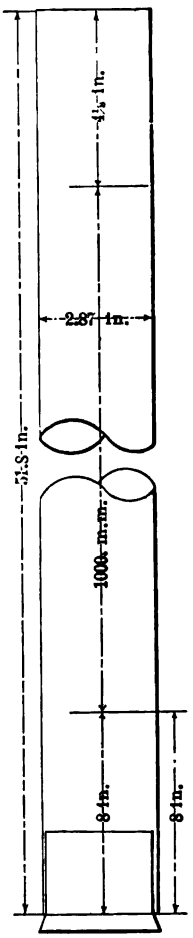

FIG. 2.—SORTING-TUBE FOR THE MIDDLE GROUP.

this hole the grains were dropped into the tube, after having been wet thoroughly, to exclude all air bubbles.

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Through 12.85 mm.		On 11.01 mm.		Aver. Diam., 11.93 mm.	Through 11.01 mm.		On 9.50 mm.		Aver. Diam., 10.257 mm.	Through 9.50 mm.		On 7.79 mm.		Aver. Diam., 8.64 mm.	Through 7.79 mm.		On 6.84 mm.		Aver. Diam., 7.31 mm.	Through 6.84 mm.		On 5.97 mm.		Aver. Diam., 6.40 mm.	Through 5.97 mm.		On 4.94 mm.		Aver. Diam., 5.45 mm.
Secs.	†No. of Grs.	Secs.	No. of Grs.		Secs.	No. of Grs.	Secs.	No. of Grs.		Secs.	No. of Grs.	Secs.	No. of Grs.		Secs.	No. of Grs.	Secs.	No. of Grs.		Secs.	No. of Grs.	Secs.	No. of Grs.		Secs.	No. of Grs.	Secs.	No. of Grs.	
2.0	1	2.0	0	2.0	2.0	0	2.0	0	2.0	2.0	0	2.0	0	2.0	2.6	0	2.8	1	3.2	3.0	1	3.2	1	3.4	3.2	1	3.6	1	3.8
2.2	5	2.2	0	2.2	2.2	0	2.2	0	2.2	2.2	0	2.2	0	2.2	2.8	1	3.0	4	3.4	3.2	6	3.4	4	3.6	3.4	5	3.8	5	4.0
2.4	29	2.4	4	2.4	2.4	4	2.4	0	2.4	2.4	0	2.4	0	2.4	3.0	10	3.2	10	3.6	3.4	19	3.6	9	3.8	3.6	15	4.0	15	4.2
2.6	23	2.6	23	2.6	2.6	23	2.6	2	2.6	2.6	2	2.6	2	2.6	3.2	23	3.4	21	3.8	3.6	18	3.8	14	4.0	3.8	28	4.2	11	4.4
2.8	21	2.8	28	2.8	2.8	28	2.8	16	2.8	2.8	16	2.8	16	2.8	3.4	21	3.8	28	4.0	4.0	33	4.2	17	4.4	4.0	33	4.4	18	4.6
3.0	15	3.0	25	3.0	3.0	25	3.0	30	3.0	3.0	30	3.0	30	3.0	3.6	20	4.0	7	4.2	4.2	17	4.4	6	4.6	4.2	17	4.6	14	4.8
3.2	3	3.2	9	3.2	3.2	9	3.2	28	3.2	3.2	28	3.2	28	3.2	3.8	13	4.2	7	4.4	4.4	6	4.6	5	4.8	4.4	6	4.8	8	5.0
3.4	2	3.4	5	3.4	3.4	5	3.4	14	3.4	3.4	14	3.4	14	3.4	4.0	4	4.4	7	4.6	4.6	5	4.8	6	5.0	4.6	5	5.0	6	5.2
3.6	0	3.6	3	3.6	3.6	3	3.6	8	3.6	3.6	8	3.6	8	3.6	4.2	5	4.6	3	4.8	4.8	6	5.0	1	5.2	4.8	6	5.2	4	5.4
3.8	0	3.8	1	3.8	3.8	2	3.8	2	3.8	3.8	2	3.8	2	3.8	4.4	1	4.8	2	5.0	5.0	1	5.2	3	5.4	5.0	1	5.4	6	5.6
4.0	1	4.0	2	4.0	4.0	2	4.0	0	4.0	4.0	0	4.0	0	4.0	4.6	0	5.0	0	5.2	5.2	2	5.4	1	5.6	5.2	3	5.6	3	5.8
*2.67	‡100	2.91	100	3.14	3.14	100	3.14	100	3.14	3.14	100	3.14	100	3.14	4.8	2	5.2	2	5.4	5.4	1	5.6	2	5.8	5.4	1	5.8	1	6.0
															3.51	100	3.80	100	4.10	5.6	2	5.8	1	6.0	5.6	2	6.0	0	6.2
																				5.8	1	6.0	1	6.2	5.8	1	6.2	1	6.4
																				6.0	1	6.2	0	6.4	6.0	0	6.4	1	6.6
																				6.2	0	6.4	1	6.6	6.2	1	6.6	0	6.8
																				6.4	1	6.6	0	6.8	6.4	0	6.8	1	7.0
																				6.6	0	6.8	1	7.0	6.6	1	7.0	0	7.2
																				6.8	1	7.0	0	7.2	6.8	0	7.2	1	7.4
																				7.0	0	7.2	1	7.4	7.0	1	7.4	0	7.6
																				7.2	1	7.4	0	7.6	7.2	0	7.6	1	7.8
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																									31.8	1	32.2	0	32.4
																									32.0	0	32.4	1	32.6
																									32.2	1	32.6	0	32.8
																									32.4	0	32.8	1	33.0
																									32.6	1	33.0	0	33.2
																									32.8	0	33.2	1	33.4
																									33.0	1	33.4	0	33.6
																									33.2	0	33.6	1	33.8
																									33.4	1	33.8	0	34.0
																									33.6	0	34.0	1	34.2
																									33.8	1	34.2	0	34.4
																									34.0	0	34.4	1	34.6
																									34.2	1	34.6	0	34.8
																									34.4	0	34.8	1	35.0
																									34.6	1	35.0	0	35.2
																									34.8	0	35.2	1	35.4
																									35.0	1	35.4	0	35.6
																									35.2	0	35.6	1	35.8
																									35.4	1	35.8	0	36.0
																									35.6	0	36.0	1	36.2
																									35.8	1	36.2	0	36.4
																									36.0	0	36.4	1	36.6
																									36.2	1	36.6	0	36.8
																									36.4	0	36.8	1	37.0
																									36.6	1	37.0	0	37.2
																									36.8	0	37.2	1	37.4
																									37.0	1	37.4	0	37.6
																									37.2	0	37.6	1	37.8
																									37.4	1	37.8	0	38.0
																									37.6	0	38.0	1	38.2
																									37.8	1	38.2	0	38.4
																									38.0	0	38.4	1	38.6
																									38.2	1	38.6	0	38.8
																									38.4	0	38.8	1	39.0
																									38.6	1	39.0	0	39.2
																									38.8	0	39.2	1	39.4
																									39.0	1	39.4	0	39.6
																									39.2	0	39.6	1	39.8
																									39.4	1	39.8	0	40.0
																									39.6	0	40.0	1	40.2
																									39.8	1	40.2	0	40.4
																									40.0	0	40.4	1	40.6
																									40.2	1	40.6	0	40.8
																									40.4	0	40.8	1	41.0
																									40.6	1	41.0	0	41.2
																									40.8	0	41.2	1	41.4
																									41.0	1	41.4	0	41.6
																									41.2	0	41.6	1	41.8
																									41.4	1	41.8	0	42.0
																									41.6	0	42.0	1	42.2
																									41.8	1	42.2	0	42.4
																									42.0	0	42.4	1	42.6
																									42.2	1	42.6	0	42.8
																									42.4	0	42.8	1	43.0
																									42.6	1	43.0	0	43.2
																									42.8	0	43.2	1	43.4
																									43.0	1	43.4	0	43.6
																									43.2	0	43.6	1	43.8
																									43.4	1	43.8	0	44.0
																									43.6	0	44.0	1	44.2
																									43.8	1	44.2	0	44.4
																									44.0	0	44.4	1	44.6
																									44.2	1	44.6	0	44.8
																									44.4	0	44.8	1	45.0
																									44.6	1	45.0	0	45.2
																									44.8	0	45.2	1	45.4
																									45.0	1	45.4	0	45.6
																									45.2	0	45.6	1	45.8
																									45.4	1	45.8	0	46.0
																									45.6	0	46.0	1	46.2
																									45.8	1	46.2	0	46.4
																									46.0	0	46.4	1	46.6
																									46.2	1	46.6	0	46.8
																									46.4	0	46.8	1	47.0
																									46.6	1	47.0	0	47.2
																									46.8	0	47.2	1	47.4
																									47.0	1	47.4	0	47.6
																									47.2	0	47.6	1	47.8
																									47.4	1	47.8	0	48.0
																									47.6	0	48.0	1	48.2
																									47.8	1	48.2	0	48.4
																									48.0	0	48.4	1	48.6
																									48.2	1	48.6	0	48.8
																									48.4	0	48.8	1	49.0
																									48.6	1	49.0	0	49.2
																									48.8	0	49.2	1	49.4
																									49.0	1	49.4	0	49.6
																									49.2	0	49.6	1	49.8
																									49.4	1	49.8	0	50.0
																									49.6	0	50.0	1	50.2
																									49.8	1	50.2	0	50.4
																									50.0	0	50.4	1	50.6
																									50.2	1	50.6	0	50.8
																									50.4	0	50.8	1	51.0
																									50.6	1	51.0	0	51.2
																									50.8	0	51.2	1	51.4
																									51.0	1	51.4	0	51.6
																									51.2	0	51.6	1	51.8
																									51.4	1	51.8	0	52.0
																									51.6	0	52.0	1	52.2
																									51.8	1	52.2	0	52.4
																									52.0	0	52.4	1	52.6
																									52.2	1	52.6	0	52.8
																									52.4	0	52.8	1	53.0
																									52.6	1	53.0	0	53.2
																									52.8	0	53.2	1	53.4
																									53.0	1	53.4	0	53.6
																									53.2	0	53.6	1	53.8
																									53.4	1	53.8	0	54.0
																									53.6	0	54.0	1	54.2
																									53.8	1	54.2	0	54.4
																									54.0	0	54.4	1	54.6
																									54.2	1	54.6	0	54.8
																									54.4	0	54.8	1	55.0
																									54.6	1	55.0	0	55.2
																									54.8	0	55.2	1	55.4
																									55.0	1	55.4	0	55.6
																									55.2	0	55.6	1	55.8
																									55.4	1	55.8	0	56.0
																									55.6	0	56.0	1	56.2

TABLE IV.—*Warren and Nagel's Seconds of Time*

Through 12.85 mm. On 11.01 mm. Aver. Diam., 11.93 mm.		Through 11.01 mm. On 9.50 mm. Aver. Diam., 10.257 mm.		Through 9.50 mm. On 7.79 mm. Aver. Diam., 8.64 mm.		Through 7.79 mm. On 6.84 mm. Aver. Diam., 7.31 mm.		Through 6.84 mm. On 5.97 mm. Aver. Diam., 6.40 mm.		Through 5.97 mm. On 4.94 mm. Aver. Diam., 5.45 mm.	
Secs.	No. of Grs.	Secs.	No. of Grs.	Secs.	No. of Grs.	Secs.	No. of Grs.	Secs.	No. of Grs.	Secs.	No. of Grs.
5.4	2	5.4	1	5.4	1	5.4	1	6.4	1	7.4	1
5.6	2	5.6	1	5.6	1	5.6	0	6.6	0	7.6	0
5.8	6	5.8	3	5.8	1	5.8	0	6.8	0	7.8	0
6.0	12	6.0	2	6.0	0	6.0	0	7.0	1	8.0	3
6.2	13	6.2	2	6.2	1	6.2	0	7.2	0	8.2	3
6.4	12	6.4	8	6.4	2	6.4	0	7.4	2	8.4	1
6.6	6	6.6	3	6.6	3	6.6	0	7.6	1	8.6	1
6.8	7	6.8	6	6.8	11	6.8	2	7.8	4	8.8	5
7.0	16	7.0	15	7.0	10	7.0	1	8.0	6	9.0	8
7.2	2	7.2	8	7.2	6	7.2	1	8.2	8	9.2	3
7.4	3	7.4	11	7.4	7	7.4	5	8.4	3	9.4	7
7.6	5	7.6	6	7.6	5	7.6	10	8.6	6	9.6	3
7.8	5	7.8	4	7.8	5	7.8	5	8.8	5	9.8	9
8.0	2	8.0	13	8.0	11	8.0	9	9.0	7	10.0	7
8.2	3	8.2	2	8.2	6	8.2	3	9.2	4	10.2	9
8.4	0	8.4	3	8.4	5	8.4	8	9.4	9	10.4	3
8.6	0	8.6	1	8.6	5	8.6	1	9.6	4	10.6	1
8.8	0	8.8	3	8.8	4	8.8	13	9.8	8	10.8	1
9.0	1	9.0	2	9.0	3	9.0	3	10.0	3	11.0	4
9.2	0	9.2	2	9.2	1	9.2	1	10.2	2	11.2	4
9.4	0	9.4	2	9.4	2	9.4	2	10.4	2	11.4	5
9.6	2	9.6	0	9.6	1	9.6	8	10.6	4	11.6	1
9.8	0	9.8	0	9.8	1	9.8	5	10.8	2	11.8	4
10.0	1	10.0	0	10.0	1	10.0	5	11.0	4	12.0	0
6.77	100	10.2	0	10.2	1	10.2	1	11.2	5	12.2	1
		10.4	0	10.4	2	10.4	2	11.4	4	12.4	5
		10.6	1	10.6	1	10.6	0	11.6	0	12.6	0
		10.8	1	10.8	1	10.8	2	11.8	1	12.8	0
		11.0	0	11.0	1	11.0	1	12.0	3	13.0	3
		7.43	100	11.2	0	11.2	5	12.6	0	13.2	2
				11.4	0	11.4	1	13.0	0	13.4	0
				11.6	0	11.6	2	14.0	0	13.6	1
				11.8	1	11.8	2	14.2	0	13.8	1
				12.0	0	12.0	1	14.8	1	14.0	0
				7.89	100	8.93	100	9.46	100	14.2	1
										14.4	0
										14.6	1
										14.8	0
										15.0	0
										15.2	1
										15.4	0
										15.6	0
										16.0	1
										10.45	100

The 100 grains for the test were taken wholly at random by spreading the group out on a glass plate in a long narrow row from which alternate inches were taken and the others rejected. This was repeated until the quantity was nearly reduced to 100 grains. This was again spread out in a long narrow row and the grains were taken from the end for the test until the row was almost used up. Grains in the galena set which evidently included blende or quartz were not accepted.

The timing was done with great care by two observers, one

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for Quartz Grains to Drop through 2800. *mm.*

Through 4.94 mm. On 4.21 mm. Aver. Diam., 4.57 mm.		Through 4.21 mm. On 4.12 mm. Aver. Diam., 4.17 mm.		Through 4.12 mm. On 2.825 mm. Aver. Diam., 3.47 mm.		Through 2.825 mm. On 2.820 mm. Aver. Diam., 2.823 mm.		Through 2.820 mm. On 2.055 mm. Aver. Diam., 2.437 mm.	
Secs.	No. of Grs.	Secs.	No. of Grs.	Secs.	No. of Grs.	Secs.	No. of Grs.	Secs.	No. of Grs.
7.8	1	7.8	1	8.8	1	10.6	1	10.8	1
8.0	0	8.0	0	9.0	0	10.8	0	11.6	1
8.2	0	8.2	0	9.2	2	11.0	0	11.8	1
8.4	1	8.4	0	9.4	0	11.4	1	12.0	1
8.6	1	8.6	0	9.6	1	11.8	2	12.4	2
8.8	1	8.8	0	9.8	1	12.0	4	12.8	0
9.0	6	9.0	3	10.0	2	12.2	2	13.0	4
9.2	2	9.2	2	10.2	3	12.4	4	13.2	2
9.4	1	9.4	0	10.4	2	12.6	3	13.4	1
9.6	3	9.6	3	10.6	1	12.8	2	13.6	1
9.8	4	9.8	3	10.8	4	13.0	5	13.8	1
10.0	11	10.0	5	11.0	2	13.2	4	14.0	5
10.2	7	10.2	6	11.2	2	13.4	3	14.2	3
10.4	2	10.4	5	11.4	3	13.6	0	14.4	3
10.6	6	10.6	4	11.6	1	13.8	6	14.6	4
10.8	6	10.8	5	11.8	6	14.0	2	14.8	3
11.0	7	11.0	6	12.0	7	14.2	4	15.0	7
11.2	2	11.2	1	12.2	0	14.4	5	15.2	1
11.4	4	11.4	1	12.4	6	14.6	3	15.4	1
11.6	1	11.6	3	12.6	1	14.8	5	15.6	5
11.8	5	11.8	4	12.8	3	15.0	4	15.8	3
12.0	2	12.0	9	13.0	3	15.2	2	16.0	5
12.2	0	12.2	0	13.2	0	15.4	2	16.2	7
12.4	1	12.4	2	13.4	3	15.6	2	16.4	2
12.6	1	12.6	0	13.6	4	15.8	1	16.6	1
12.8	3	12.8	3	13.8	2	16.0	2	16.8	2
13.0	5	13.0	5	14.0	1	16.2	0	17.0	0
13.2	1	13.2	2	14.2	1	16.4	1	17.2	1
13.4	3	13.4	2	14.4	2	16.6	3	17.4	1
13.6	1	13.6	0	14.6	3	16.8	1	17.8	2
13.8	3	13.8	3	14.8	2	17.2	1	18.0	1
14.0	0	14.0	3	15.0	2	17.4	1	18.2	2
14.2	0	14.2	0	15.2	1	17.8	0	18.4	3
14.4	0	14.4	1	15.4	1	18.0	2	18.6	1
14.6	1	14.6	3	15.6	1	18.4	4	19.0	1
14.8	0	14.8	3	15.8	4	18.6	1	19.2	2
15.0	0	15.0	4	16.0	2	18.8	2	19.4	2
15.2	1	15.2	0	16.2	2	19.0	2	19.6	2
15.4	0	15.4	0	16.4	1	19.2	1	19.8	2
15.6	1	15.6	1	16.6	1	19.4	1	20.0	1
15.8	1	15.8	1	16.8	0	19.6	2	20.2	1
16.0	1	16.0	1	17.0	3	19.8	2	20.4	1
16.2	1	16.4	1	17.2	0	20.0	2	20.6	1
17.0	1	16.6	1	17.4	2	20.2	1	20.8	1
17.4	1	16.8	2	17.6	0	20.8	1	21.0	1
18.0	1	17.0	0	17.8	1	21.0	1	21.2	1
11.36	100	17.8	1	18.0	0	21.4	1	21.4	1
		18.0	0	13.03	100	21.8	1	21.6	1
		12.08	100			15.31	100	22.0	2
								22.8	1
								23.2	1
								16.37	100

at the upper sight, the other at the lower sight. The upper observer dropped a grain through the hole in the zinc top, and when the grain passed the sight called to the man below, who started his stop-watch. The latter stopped his watch when the grain passed the lower sight. The figures recorded in Tables III. and IV. represent the time elapsed.

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Tables III. and IV. show the number of galena and quartz grains that fell 2800. mm. at each fifth of a second between the fastest and the slowest speed for each size. The average velocity given at the foot of each column is that for the whole 100 grains. The final summing up of these values is given in Table XI.

A series of tests was made to see if the groups taken at random really covered the field. For this purpose very flat, slow moving grains were selected. Occasionally a flat grain was found that fell more slowly than any in the table, but the number was too small to affect the results.

A test was made to ascertain the error in timing. We will use the word "lag" to represent the time elapsed between the passing of the particle and the record on the stop-watch. The lag of seeing and recording on the stop-watch by the same observer we will call single lag, the acts of a single brain. The lag of seeing and reporting by one observer to the other who hears and records on the stop-watch we will call "double lag," the acts of two brains. When the upper observer recorded both events on the watch he made single lag at the start and double lag at the finish. The time was, therefore, too long by a single lag. When the lower observer held the watch, he made double lag at the top and single lag at the bottom. This time was too short by a single lag. The average difference on 42 tests showed that the time was 0.67 sec. (double lag) longer when the watch was at the top than when it was at the bottom. When, therefore, the watch was held at the bottom, as was the case in the tests, the time of settling recorded on the watch was 0.33 sec. (one-half of 0.67 sec.) too short. The figures given in Tables III. and IV. show the actual record. In Table XI. the error has been corrected.

A separate test was made to record, together with the velocities, the shapes of the grains. A group of galena grains and a group of quartz grains passing through the screen 4.125 mm. and resting on the screen 2.825 mm. were taken for this purpose. (Tables V. and VI.) It will be noted that the slower grains are nearly all long or flat and that the faster grains are nearly all cubical. The slightly slower speed of the grains in this test as compared with those of the former test (Tables III.

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TABLE V.—*Galena of* 3.47 *mm. Average Diameter Dropped* 2800. *mm.*

Timer.	Seconds.	Weight.	Description.	Timer.	Seconds.	Weight.	Description.
N.	4.0	0.38	Massive.	W.	5.2	0.16	Irregular.
N.	4.0	0.37	Massive, irregular	W.	5.2	0.14	Cubical.
W.	4.2	0.55	Cubical.	N.	5.4	0.50	Irregular, flat.
N.	4.2	0.46	Cubical.	W.	5.4	0.28	Irregular.
N.	4.2	0.40	Massive.	N.	5.4	0.26	Cubical.
W.	4.2	0.26	Massive.	W.	5.4	0.25	Square, long.
N.	4.4	0.49	Cubical.	W.	5.4	0.22	Irregular.
N.	4.4	0.45	Long, irregular.	W.	5.4	0.21	Cubical.
N.	4.4	0.40	Cubical.	N.	5.4	0.19	Massive.
N.	4.4	0.33	Massive.	N.	5.4	0.19	Long.
W.	4.4	0.31	Cubical	N.	5.4	0.16	Massive.
N.	4.4	0.29	Cubical.	N.	5.4	0.18	Massive.
N.	4.4	0.28	Massive.	N.	5.4	0.15	Cubical.
N.	4.4	0.27	Long, irregular.	W.	5.6	0.45	Long, flat.
N.	4.4	0.27	Cubical.	N.	5.6	0.36	Long, irregular.
N.	4.4	0.26	Cubical.	W.	5.6	0.28	Massive, flat.
N.	4.4	0.20	Cubical.	N.	5.6	0.26	Long, irregular.
N.	4.4	0.17	Cubical.	W.	5.6	0.18	Massive.
N.	4.6	0.44	Cubical.	N.	5.8	0.62	Square, long.
N.	4.6	0.39	Massive.	W.	5.8	0.43	Long.
N.	4.6	0.38	Massive.	W.	5.8	0.26	Massive, irregular.
W.	4.6	0.38	Irregular.	N.	5.8	0.19	Long, flat.
N.	4.6	0.33	Massive.	W.	5.8	0.17	Massive, irregular.
N.	4.6	0.33	Long, massive.	W.	5.8	0.16	Cubical.
N.	4.6	0.33	Square, flat.	W.	5.8	0.13	Massive.
W.	4.6	0.18	Cubical.	W.	5.8	0.12	Cubical.
W.	4.6	0.15	Ordinary.	W.	6.0	0.43	Long, massive.
N.	4.8	0.67	Long, flat.	W.	6.0	0.26	Cubical.
N.	4.8	0.42	Irregular.	W.	6.0	0.21	Irregular.
W.	4.8	0.38	Massive.	N.	6.0	0.21	Cubical.
N.	4.8	0.33	Irregular.	W.	6.0	0.15	Irregular.
N.	5.0	0.36	Irregular.	N.	6.2	0.30	Long, irregular.
W.	5.0	0.35	Massive.	W.	6.2	0.26	Square, flat.
W.	5.0	0.30	Cubical.	N.	6.2	0.22	Irregular, massive.
N.	5.0	0.28	Massive.	W.	6.2	0.14	Cubical.
N.	5.0	0.27	Irregular.	W.	6.2	0.14	Cubical.
N.	5.0	0.18	Cubical.	W.	6.2	0.12	Square, flat.
N.	5.0	0.21	Cubical.	W.	6.4	0.31	Long, irregular.
N.	5.0	0.18	Cubical.	W.	6.4	0.30	Long, massive.
W.	5.0	0.18	Cubical.	N.	6.4	0.10	Irregular.
W.	5.2	0.63	Long, irregular.	W.	6.6	0.23	Massive, flat.
W.	5.2	0.43	Square, long.	W.	7.0	0.21	Flat.
N.	5.2	0.34	Long, massive.	W.	7.0	0.18	Square, flat.
N.	5.2	0.30	Long, flat.	N.	7.0	0.17	Wide, flat.
W.	5.2	0.30	Irregular.	W.	7.0	0.15	Flat, irregular.
W.	5.2	0.28	Cubical.	W.	7.2	0.20	Long, flat.
W.	5.2	0.27	Cubical.	N.	7.4	0.22	Long, flat.
N.	5.2	0.27	Irregular.	W.	7.4	0.19	Square, flat.
W.	5.2	0.26	Cubical.	W.	7.4	0.17	Wide, flat.
N.	5.2	0.24	Massive.	N.	7.4	0.15	Flat.
				N.	7.8	0.09	Irregular, flat.
				W.	8.0	0.10	Cubical.
				N.	8.0	0.26	Flat.
				N.	8.6	0.15	Irregular, flat.
				W.	9.0	0.16	Wide, flat.
					5.51	0.27	Average.

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TABLE VI.—*Quartz of* 3.47 *mm. Average Diameter Dropped* 2800. *mm.*

Timer.	Seconds.	Weight.	Description.	Timer.	Seconds.	Weight.	Description.
N.	8.8	0.07	Ordinary.	W.	13.0	0.08	Irregular.
W.	9.2	0.08	Ordinary.	W.	13.0	0.07	Massive.
W.	9.2	0.08	Massive.	W.	13.0	0.07	Massive.
N.	9.6	0.11	Long.	N.	13.0	0.06	Ordinary.
W.	9.6	0.08	Massive.	N.	13.0	0.05	Flat, spinner.
N.	9.8	0.11	Massive.	N.	13.0	0.04	Ordinary.
N.	10.0	0.14	Long.	W.	13.2	0.07	Massive.
N.	10.0	0.07	Ordinary.	W.	13.2	0.06	Massive.
W.	10.2	0.09	Massive.	N.	13.4	0.05	Irregular.
W.	10.4	0.09	Irregular.	W.	13.6	0.07	Irregular, flat.
W.	10.4	0.07	Massive.	W.	13.6	0.05	Massive.
N.	10.4	0.06	Ordinary.	W.	13.6	0.05	Lump.
N.	10.6	0.07	Ordinary.	W.	13.8	0.15	Long, spinner.
N.	10.6	0.04	Ordinary.	N.	13.8	0.10	Flat, massive, spinner.
W.	10.8	0.09	Irregular.	W.	13.8	0.08	Long.
W.	10.8	0.09	Massive.	W.	14.0	0.13	Long.
N.	11.0	0.12	Ordinary.	W.	14.0	0.09	Flattish.
N.	11.0	0.11	Massive.	N.	14.0	0.08	Long.
N.	11.0	0.11	Massive.	N.	14.0	0.08	Long.
W.	11.0	0.07	Ordinary.	W.	14.0	0.06	Massive.
N.	11.2	0.14	Long, irregular, twister.	W.	14.0	0.05	Irregular.
N.	11.2	0.06	Ordinary.	W.	14.0	0.05	Massive.
N.	11.2	0.06	Ordinary.	N.	14.0	0.05	Irregular, flat.
N.	11.2	0.05	Long.	N.	14.0	0.04	Irregular.
W.	11.4	0.10	Irregular.	W.	14.2	0.08	Flat.
N.	11.4	0.08	Irregular.	N.	14.2	0.04	Ordinary.
N.	11.4	0.05	Ordinary.	N.	14.2	0.03	Ordinary.
N.	11.4	0.05	Massive.	N.	14.4	0.10	Long.
W.	11.8	0.14	Irregular.	W.	14.8	0.18	Long, flat.
N.	11.8	0.12	Flat, massive.	W.	14.8	0.07	Flat, wabbler.
N.	11.8	0.09	Ordinary.	N.	14.8	0.06	Long, flat.
N.	11.8	0.10	Long.	W.	14.8	0.04	Massive.
N.	11.8	0.08	Irregular.	W.	15.0	0.08	Long.
W.	11.8	0.08	Irregular.	W.	15.0	0.05	Massive.
W.	11.8	0.07	Ordinary.	N.	15.0	0.04	Ordinary.
W.	11.8	0.06	Ordinary.	W.	15.0	0.03	Flat.
W.	12.0	0.11	Flat.	W.	15.2	0.03	Irregular.
W.	12.0	0.07	Massive.	W.	15.4	0.07	Flat, spinner.
N.	12.0	0.06	Ordinary.	W.	15.8	0.04	Irregular, flat.
N.	12.0	0.06	Ordinary.	W.	16.0	0.06	Flat, wabbler.
W.	12.2	0.08	Massive.	N.	16.2	0.12	Long, massive, flat.
W.	12.2	0.08	Massive.	N.	16.4	0.07	Long, flat.
W.	12.2	0.06	Massive.	N.	17.2	0.08	Flat, wabbler.
N.	12.2	0.05	Ordinary.	W.	17.4	0.04	Flat.
W.	12.4	0.13	Irregular.	W.	18.0	0.03	Long.
W.	12.4	0.10	Long.	N.	18.2	0.05	Flat, spinner.
W.	12.4	0.07	Flattish.	N.	18.4	0.04	Long, flat, spinner.
				N.	18.6	0.05	Flat, twister.
				W.	20.0	0.06	Long, thin, wabbler.
				W.	20.0	0.05	Irregular, flat, spinner.
				N.	20.8	0.06	Long, twister.
				N.	21.0	0.03	Flat.
				W.	21.2	0.03	Flat.
				N.	22.2	0.03	Very flat.
				N.	22.6	0.05	Long, flat, spinner.
					13.5	0.07	Average.

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and IV.) is supposed to be due to the fact that these were all selected grains, while those of the former were taken at random.

The behavior of the grains in falling is thus defined: "wabbler," one that has a zigzag course; "twister," one that descends in a helical path; "spinner," one that descends in the center but rotates rapidly as it goes. The rest of the grains dropped approximately in a straight line.

The letters "N" and "W," in the first column of Tables V. and VI., stand for Nagel and Warren.

III. The Second, or the Middle, Section of the Field.

The next section of the field was covered by Messrs. G. A. Barnaby and Ralph Hayden. Their sieves are given in Table VII. The sizes were measured by Mr. Robert F. Manahan by microscope and micrometer scale, adopting the mean of a number of observations.

Table VII.—*Double Rittinger Sieve-Scale Used by Barnaby and Hayden.*

	Diameter of Grains, mm.		
Double Rittinger.	Through mm.	On mm.	Average mm.
2.38	2.49	2.06	2.28
2.00	2.06	1.63	1.85
1.681	1.63	1.46	1.55
1.414	1.46	1.27	1.37
1.189	1.27	1.10	1.19
1.000	1.10	0.97	1.04
0.841	0.97	0.84	0.91
0.707	0.84	0.68	0.76
0.595	0.68	0.57	0.63
0.500	0.57	0.45	0.51
0.4205	0.45	0.36	0.41
0.3535	0.36	0.28	0.32
0.25			

It will be noted that the sieves do not correspond exactly with the double Rittinger scale, but the intervals between the sizes in the Tyler sieves which were used are so nearly the same as the double Rittinger that the purpose of the investigation was fully met by these sieves.

The test-tube adopted for these smaller grains was a glass tube, Fig. 2, 51.8 in. high and $2\frac{1}{8}$ in. in diameter, on which two courses were laid out, one 1 m. long, the other 0.5 m. long.

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The distance from the top of the tube to the first sight for both courses was $4\frac{1}{8}$ inches.

All the galena grains and the quartz grains between 2.49 mm. and 0.97 mm. were dropped through the 1-m. course. The quartz grains from 0.97 to 0.28 were dropped through the 0.5 m., and the figures for these last results were multiplied by 2 in order to place them all on a uniform basis in the table. The results of these tests are given in Tables VIII. and IX., and the final summing-up of them in Table XI. The figures given at the foot of the column of seconds are the average seconds for the 100 grains. As the same observer timed the start and finish in this field, there is no error due to lag to be corrected in Table XI.

Warren drew curves of distribution of grains according to the velocity, which might have some significance, and found a bunch of grains among the quicker speeds and another among the slower. It may be that something can be found here; but Barnaby's results differ in showing for the coarser sizes of his crushing that the grains are largely bunched at one place, while for the smaller sizes they may be bunched at one or more places. I do not feel that anything sufficiently definite has been found upon which to base conclusions.

IV. The Third, or Finest, Section of the Field.

The tests of Barnaby and Hayden carry the measures of velocity of settling down to the limit of sifting—namely, 0.28-mm. grains.

Mr. E. S. Bardwell took up the work at this point, using the elutriation method. This consists of stirring up the grains in water, settling for a specified time and distance, and decanting the water with the suspended grains; then measuring the diameters of the decanted grains by a microscope and micrometer; and finally repeating the operation for a complete series.

Before proceeding with the determination of the velocity, it was necessary to crush samples of galena and quartz. Bardwell put 500 g. of each through the Hendrie and Bolthoff sample-grinder, starting with grains 2.83 mm. in diameter and bringing them down to pass through a 0.45-mm. screen. This was chosen a little larger than the 0.28-mm. screen of Barnaby and

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TABLE VIII.—*Seconds of Time for Galena Grains to Drop through 1 Meter.*

Through 2.49 mm.		Through 2.06 mm.		Through 1.68 mm.		Through 1.46 mm.		Through 1.27 mm.		Through 1.10 mm.		Through 0.97 mm.		Through 0.84 mm.		Through 0.68 mm.		Through 0.57 mm.		Through 0.45 mm.		Through 0.36 mm.	
Secs.	No. of Grs.	Secs.	No. of Grs.	Secs.	No. of Grs.	Secs.	No. of Grs.	Secs.	No. of Grs.	Secs.	No. of Grs.	Secs.	No. of Grs.	Secs.	No. of Grs.	Secs.	No. of Grs.	Secs.	No. of Grs.	Secs.	No. of Grs.	Secs.	No. of Grs.
1.8	12	1.8	1	2.4	7	2.6	5	2.8	3	2.8	1	3.2	2	3.6	6	3.8	1	4.2	1	5.6	1	6.6	1
2.0	25	2.0	1	2.6	16	2.8	14	3.0	12	3.0	8	3.4	5	3.8	7	4.0	2	4.4	1	6.2	2	6.8	1
2.2	25	2.2	12	2.8	28	3.0	20	3.2	14	3.2	10	3.4	10	4.0	10	4.2	6	4.6	3	6.4	3	7.0	2
2.4	18	2.4	21	3.0	20	3.2	16	3.4	17	3.4	12	3.8	14	4.2	5	4.4	10	4.8	3	6.6	3	7.2	5
2.6	10	2.6	20	3.2	6	3.4	14	3.6	12	3.6	16	4.0	9	4.4	15	4.6	8	5.0	6	6.8	6	7.4	2
2.8	4	2.6	18	3.4	9	3.6	9	3.8	12	3.8	11	4.2	11	4.6	12	4.8	11	5.2	7	7.0	7	7.6	5
3.0	2	2.8	14	3.6	5	3.8	7	4.0	9	4.0	14	4.4	11	4.8	11	5.0	14	5.4	7	7.2	10	7.8	2
3.2	2	3.2	5	3.8	4	4.0	3	4.2	5	4.2	4	4.6	7	5.0	11	5.2	8	5.6	7	7.4	7	8.0	4
3.4	1	3.4	4	4.0	2	4.2	4	4.4	4	4.4	8	4.8	6	5.2	5	5.4	11	5.8	5	7.6	10	8.2	5
3.6	1	3.6	2	4.2	2	4.4	0	4.4	4	4.6	9	5.0	10	5.2	5	5.6	5	6.0	7	7.8	4	8.4	4
2.264	100	3.8	0	4.4	0	4.6	4	4.8	3	4.8	2	5.2	5	5.4	1	5.8	3	6.2	5	8.0	3	8.6	4
		4.0	0	4.6	1	4.8	1	5.0	1	5.0	3	5.4	3	5.6	2	6.0	2	6.4	3	8.2	2	8.8	5
		4.2	1	4.8	1	5.0	1	5.2	1	5.2	4	5.6	2	5.8	2	6.2	0	6.6	1	8.4	2	9.0	5
		4.4	1	5.0	0	5.2	0	5.4	1	5.4	1	5.8	2	6.0	2	6.4	3	6.8	1	8.6	3	9.2	1
		2.704	100	5.2	0	5.4	0	5.6	2	5.6	1	6.2	0	6.4	0	6.6	1	7.0	2	8.8	3	9.4	4
				5.4	1	5.6	0	3.700	100	5.8	0	6.4	0	6.6	2	6.8	2	7.2	3	9.0	1	9.6	4
				3.026	100	5.8	1			6.0	0	6.6	1	6.8	1	7.0	3	7.4	3	9.2	2	9.8	5
						6.0	1			6.2	1	6.8	1	7.6	1	7.2	1	7.6	4	9.4	6	10.0	5
						3.388	100			3.960	100	4.398	100	7.6	1	5.182	100	7.8	1	9.6	3	10.2	6
														9.2	1			8.0	2	9.8	2	10.4	3
														4.814	100			8.2	0	10.0	3	10.6	3
																		8.4	3	10.2	1	10.8	4
																		8.6	1	10.4	2	11.0	2
																		9.2	1	10.6	1	11.2	1
																		12.2	1	11.6	2	11.4	2
																		6.234	100	13.4	1	11.8	1
																				7.982	100	12.0	1
																						12.4	1
																						12.8	2
																						13.0	2
																						13.4	1
																						13.6	2
																						13.8	1
																						14.4	1
																						15.0	1
																						18.2	1
																						9.694	100

Through 0.36 mm.		Through 0.28 mm.		Aver. Diam., 0.82 mm.	
Secs.	No. of Grs.	Secs.	No. of Grs.	Secs.	No. of Grs.
6.6	1				
6.8	1				
7.0	2				
7.2	5				
7.4	2				
7.6	5				
7.8	2				
8.0	4				
8.2	5				
8.4	4				
8.6	4				
8.8	5				
9.0	5				
9.2	1				
9.4	4				
9.6	4				
9.8	4				
10.0	5				
10.2	5				
10.4	6				
10.6	3				
10.8	4				
11.0	2				
11.2	1				
11.4	2				
11.8	2				
12.0	1				
12.4	1				
12.8	1				
18.0	2				
13.4	1				
13.6	2				
13.8	1				
14.4	1				
15.0	1				
18.2	1				
9.694	100				

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TABLE IX.—*Seconds of Time for Quartz*

Through 2.49 mm. On 2.06 mm. Aver. Diam., 2.28 mm.		Through 2.06 mm. On 1.63 mm. Aver. Diam., 1.85 mm.		Through 1.63 mm. On 1.46 mm. Aver. Diam., 1.55 mm.		Through 1.46 mm. On 1.27 mm, Aver. Diam., 1.37 mm.		Through 1.27 mm. On 1.10 mm. Aver. Diam., 1.19 mm.		Through 1.10 mm. On 0.97 mm. Aver. Diam., 1.04 mm.	
Secs.	No. of Grs.	Secs.	No. of Grs.	Secs.	No. of Grs.	Secs.	No. of Grs.	Secs.	No. of Grs.	Secs.	No. of Grs.
4.2	2	4.4	2	5.2	1	5.8	1	6.4	2	6.8	1
4.4	2	4.6	0	5.4	0	6.0	0	6.6	1	7.0	0
4.6	2	4.8	1	5.6	1	6.2	0	6.8	0	7.2	0
4.8	7	5.0	3	5.8	4	6.4	4	7.0	2	7.4	2
5.0	11	5.2	3	6.0	9	6.6	4	7.2	2	7.6	1
5.2	14	5.4	6	6.2	5	6.8	5	7.4	6	7.8	0
5.4	7	5.6	3	6.4	4	7.0	4	7.6	7	8.0	3
5.6	6	5.8	5	6.6	5	7.2	8	7.8	3	8.2	1
5.8	5	6.0	10	6.8	3	7.4	4	8.0	8	8.4	6
6.0	8	6.2	10	7.0	6	7.6	6	8.2	7	8.6	5
6.2	9	6.4	8	7.2	7	7.8	3	8.4	1	8.8	4
6.4	3	6.6	5	7.4	3	8.0	6	8.6	3	9.0	1
6.6	5	6.8	9	7.6	8	8.2	7	8.8	5	9.2	2
6.8	1	7.0	4	7.8	3	8.4	7	9.0	6	9.4	5
7.0	7	7.2	4	8.0	8	8.6	7	9.2	2	9.6	5
7.2	0	7.4	4	8.2	5	8.8	4	9.4	5	9.8	6
7.4	3	7.6	4	8.4	3	9.0	3	9.6	2	10.0	6
7.6	1	7.8	0	8.6	2	9.2	3	9.8	3	10.2	4
7.8	0	8.0	5	8.8	1	9.4	2	10.0	2	10.4	5
8.0	1	8.2	1	9.0	2	9.6	4	10.2	2	10.6	5
8.2	0	8.4	2	9.2	2	9.8	1	10.4	2	10.8	5
8.4	0	8.6	0	9.4	0	10.0	3	10.6	3	11.0	0
8.6	0	8.8	4	9.6	2	10.2	3	10.8	2	11.2	4
8.8	1	9.0	0	9.8	0	10.4	2	11.0	4	11.4	3
9.0	0	9.2	0	10.0	6	10.6	2	11.2	2	11.6	0
9.2	0	9.4	1	10.2	2	10.8	0	11.4	3	11.8	1
9.4	0	9.6	2	10.4	0	11.0	1	11.6	1	12.0	1
9.6	1	9.8	0	10.6	0	11.2	1	11.8	0	12.2	5
9.8	0	10.0	0	10.8	1	11.4	0	12.0	3	12.4	0
10.0	0	10.2	0	11.0	0	11.6	1	12.2	1	12.6	0
10.2	0	10.4	0	11.6	3	11.8	1	12.4	3	12.8	1
10.4	1	10.6	0	11.8	1	12.0	1	12.6	0	13.0	6
10.6	1	10.8	1	12.0	1	12.2	1	12.8	0	13.2	2
10.8	0	11.0	0	13.4	1	14.4	1	13.0	0	13.4	2
11.0	1	11.2	1	14.6	0	8.440	100	13.2	1	13.6	1
11.2	1	11.4	0	14.8	1			13.4	1	13.8	2
5.994 (average)	100	11.6	1	7.866	100			13.6	1	14.0	0
		14.0	1					14.2	1	14.2	1
		6.830	100					14.4	1	14.4	0
								14.6	1	14.6	1
								15.2	1	14.8	0
								9.472	100	15.6	1
										17.0	1
										18.2	1
										10.596	100

Hayden in order that his work should overlap a little on theirs, and that no gap should be left between the sifted and the floated grains.

Two designs of apparatus were made by Bardwell. That for slower speeds, Fig. 3, consisted of a bottle, *a*, with the bottom cut off at *b*, a cork stopper, *c*, a discharge-pipe, *d*, a rubber tube and pinch-cock, *e*, a glass guard-tube, *f*, and an upper sight 100 mm. above the end of the discharge-tube. The

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Grains to Drop through 1 Meter.

Through 0.97 mm. On 0.84 mm. Aver. Diam., 0.91 mm.		Through 0.84 mm. On 0.68 mm. Aver. Diam., 0.76 mm.		Through 0.68 mm. On 0.57 mm. Aver. Diam., 0.63 mm.		Through 0.57 mm. On 0.45 mm. Aver. Diam., 0.51 mm.		Through 0.45 mm. On 0.36 mm. Aver. Diam., 0.41 mm.		Through 0.36 mm. On 0.28 mm. Aver. Diam., 0.32 mm.	
Secs.	No. of Grs.	Secs.	No. of Grs.	Secs.	No. of Grs.	Secs.	No. of Grs.	Secs.	No. of Grs.	Secs.	No. of Grs.
8.2	3	8.8	1	10.8	1	12.0	1	16.4	1	20.8	1
8.8	3	9.2	4	11.2	1	12.4	1	16.8	1	21.2	3
9.2	9	9.6	3	11.6	5	12.8	1	17.2	1	21.6	1
9.6	10	10.0	8	12.0	7	13.2	1	17.6	0	22.0	0
10.0	9	10.4	3	12.4	5	13.6	3	18.0	2	22.4	1
10.4	3	10.8	9	12.8	5	14.0	2	18.4	1	22.8	1
10.8	11	11.2	6	13.2	8	14.4	1	18.8	1	23.2	4
11.2	3	11.6	9	13.6	11	14.8	1	19.2	2	23.6	2
11.6	6	12.0	8	14.0	9	15.2	3	19.6	5	24.0	2
12.0	3	12.4	2	14.4	5	15.6	5	20.0	3	24.8	1
12.4	4	12.8	4	14.8	6	16.0	6	20.4	2	25.2	6
12.8	7	13.2	5	15.2	5	16.4	6	20.8	7	25.6	2
13.2	3	13.6	6	15.6	5	16.8	5	21.2	3	26.0	3
13.6	4	14.0	3	16.0	3	17.2	6	21.6	3	26.4	4
14.0	4	14.4	5	16.4	3	17.6	6	22.0	4	26.8	3
14.4	2	14.8	3	16.8	4	18.0	8	22.4	3	27.2	2
14.8	7	15.2	1	17.2	3	18.4	3	22.8	5	27.6	1
15.2	0	15.6	2	17.6	3	18.8	4	23.2	4	28.0	3
15.6	2	16.0	3	18.0	3	19.2	6	23.6	5	28.4	4
16.0	3	16.4	1	18.4	2	19.6	2	24.0	6	29.6	2
16.4	0	16.8	3	18.8	0	20.0	3	24.4	3	30.0	3
16.8	3	17.2	2	19.2	1	20.4	1	24.8	5	30.4	3
23.2	1	17.6	3	19.6	0	20.8	3	25.2	1	30.8	2
11.834	100	18.0	0	20.0	0	21.2	1	25.6	3	31.2	2
		18.4	1	20.4	1	21.6	2	26.0	2	31.6	3
		18.8	2	20.8	0	22.0	2	26.4	2	32.0	3
		19.2	0	22.4	1	22.4	3	26.8	4	32.4	4
		19.6	0	22.8	1	22.8	2	27.2	1	32.8	4
		20.0	0	26.0	1	23.2	1	27.6	0	33.6	1
		20.4	1	27.6	1	23.6	1	28.4	2	34.0	3
		20.8	1	14.840	100	24.8	1	28.8	1	34.4	4
		24.8	1			26.8	1	29.4	1	34.8	1
		13.028	100			27.2	1	29.6	1	37.6	1
						27.6	0	30.0	1	38.0	1
						28.0	2	31.2	1	38.4	1
						28.4	1	31.6	4	39.2	4
						29.6	1	32.0	3	40.8	1
						31.6	1	32.8	2	41.2	1
						36.8	1	33.6	1	41.6	2
						38.0	0	34.8	1	42.0	1
						38.4	1	37.6	2	42.4	1
						18.986	100	24.346	100	42.8	4
										44.8	1
										45.2	1
										55.6	1
										64.8	1
										31.360	100

mode of operating was as follows: The sand (30 g. for quartz, 100 g. for galena) was charged, stirred up thoroughly with a glass rod, and allowed to settle the specified time. (See Table X.) The guard-tube was then lifted out, and at the same time the discharge-cock was opened, decanting off the top water. After this had been done five times, about all the light grains had been removed from the heavy. Theoretically, 0.12 per cent. of the fine grains would remain with the coarse.

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The second apparatus, Fig. 4, consisted of a glass settling-tube, *g*, a cork stopper, a glass discharging-tube, *i*, a stopper, *r*, with handle of glass tube, *j*, and a cork stopper, *k*, an air-vent, *l*, a water-entrance, *m*, and two pinch-cocks, *pp*. The method of using this was to put in the sand; adjust the plug,

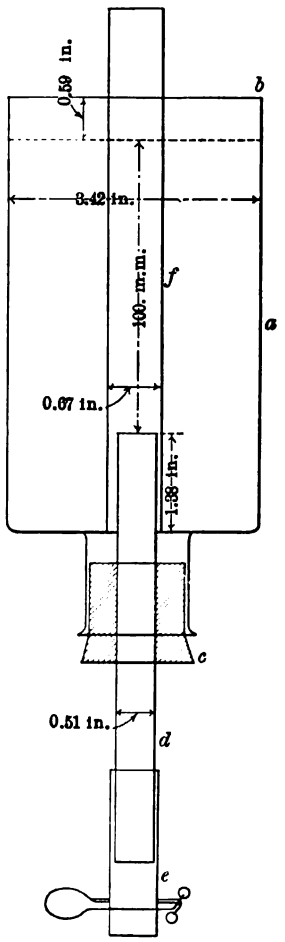

FIG. 3.—SORTING-BOTTLE FOR THE SLOWER GRAINS OF THE FINEST GROUP.

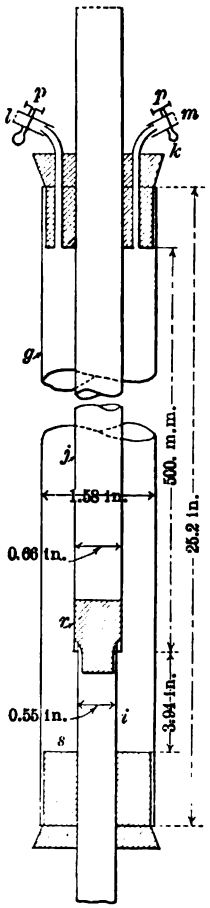

FIG. 4.—SORTING-TUBE FOR THE QUICKER GRAINS OF THE FINEST GROUP.

r, the handle, *j*, and the stopper, *k*; to admit water at *m*, and drive out air at *l*; next, to invert and mix the water and sand thoroughly together; then restore to working position. In this case the time for settling 500 mm. was recorded. When the grains had settled the required time, the cork, *k*, the

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handle, *j*, and the stopper, *r*, were withdrawn and the water with its suspended sand was discharged. The plug, *k*, the handle, *j*, and the stopper, *r*, were then replaced, the tube refilled with water, inverted to bring the grains to the top of the tube, reversed to working position, timed again and discharged. This test performed five times gave in the discharged water about all the light grains, and at the bottom, *s*, of the tube, all the heavy grains. Theoretically, 0.014 per cent. of the light grains would remain with the heavy.

All the quartz grains were tested in the first apparatus, Fig. 3, but in testing the galena grains it was found that when the time became less than 4.5 sec. it was advisable to use the second apparatus (Fig. 4), which settles 500 mm. instead of 100 millimeters.

After the settling-apparatus was designed and the mineral crushed, it was necessary to decide on a scale of time for settling which would serve to divide the grains into groups for this part of the field in the same way that the sieves did for the two coarser fields. A grain of quartz, 0.35 mm. in diameter, which settles 100 mm. in 2.432 sec., was taken as the starting-point, and the double Rittinger factor (1.189) was used to multiply the seconds of time; 2.432 sec. multiplied by 1.189 gave the second time-value, multiplied by 1.189^2 it gave the third, by 1.189^3 the fourth, and so on until 1.189^{36} gave 1082.6 sec. as the time for settling quartz 100 mm. This last gives grains small enough for the fine end of the series. Thus the complete series of times was adopted for quartz, as given in Table X. The series was made for galena in this way: A 0.36-mm. grain of galena settles 100 mm. in 0.7928 sec.; multiplied by 1.189 this gave the second time-value; by 1.189^2 gave the third, and so on until 1.189^{42} was reached and gave 1005.0 sec. as the settling-time for galena. This gives grains small enough for the small end of the series. This complete series of times was adopted for galena (Table X).

In conducting the experiments, one can keep the volumes of water of reasonable size by beginning with the small grains—that is, with those that have the slowest rate of settling. On this account the several tests were tried in the reverse order from that given in Table X.

Since the ten quickest groups of galena grains were treated

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TABLE X.—*Double Rittinger Time-Scale Used by Bardwell.*

No. of Decantation.	100 mm.		No. of Decantation.	100 mm.	
	Quartz Seconds.	Galena Seconds.		Quartz Seconds.	Galena Seconds.
1	2.4	0.8	22	94.6	30.8
2	2.9	0.9	23	112.7	36.7
3	3.5	1.12	24	134.0	43.7
4	4.1	1.34	25	159.5	52.0
5	4.9	1.59	26	190.0	62.0
6	5.8	1.9	27	226.0	73.7
7	6.9	2.3	28	269.0	87.7
8	8.2	2.7	29	320.0	104.3
9	9.8	3.2	30	381.0	124.1
10	11.7	3.8	31	454.0	147.9
11	14.0	4.5	32	540.0	176.0
12	16.6	5.4	33	643.0	209.3
13	19.8	6.4	34	765.2	249.2
14	23.5	7.7	35	910.6	296.5
15	28.0	9.1	36	1082.6	353.0
16	33.3	10 8	37	.....	420.0
17	39.7	12.9	38	.....	500.0
18	47.1	15.4	39	.....	595.0
19	56.1	18.3	40	.....	709.0
20	66.8	21.8	41	.....	844.0
21	79.5	25.9	42	.....	1005.0

with 500-mm. settling-column, their time-values, as given in Table X., had to be multiplied by five for use in the tube.

The results of these measures are given in Table XI., and with them the corresponding velocities in mm. a second. These latter values are derived from the velocities, which are given in the table in seconds for 100 millimeters.

V. DISCUSSION OF RESULTS.

In Table XI. we have the summing-up of the work of all three fields recomputed to the uniform basis of mm. a second for velocity. We have also a mathematical computation of the rate of fall and the ratio between this theoretical and the actual rate of fall.

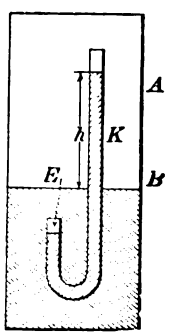

FIG. 5.—COLUMN OF WATER TO SUPPORT A CUBE OF MINERAL.

Discussing these results mathematically from the point of view taken by Rittinger, we have in Fig. 5: *A*, a jar with water in it up to *B*; *K*, a U-tube with a square section, which we will call D meters square inside section. Upon the lower end is a cube of mineral, *E*, D meters cube; and within the tube is a column, *h*, of water just high enough to balance the weight of the cube, *E*.

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By Warren and Nagel—Average velocities are of 100 observations.

Diameters of grains Millimeters.			Galena Velocities.			Comp't Vel'cty V.	Ratio Average divided by Compt f.	Quartz Velocities			Comp Vel. V.	Ratio Average divided by Comp f.
Thro mm.	On mm.	Av. mm.	Max. mm. Sec.	Min. mm. Sec.	Aver. mm. Sec.	mm. Sec.		Max. mm. Sec.	Min. mm. Sec.	Aver. mm. Sec.	mm. Sec.	
12.85	11.01	11.93	1201.	647.	934.	1235.	0.7576	489.	271.	398.	621.5	0.6325
11.01	9.51	10.26	1016.	647.	865.	1142.	0.7576	489.	252.	361.	576.	0.6269
9.51	7.79	8.65	956.	678.	810.	1052.	0.7704	489.	231.	340.	582.	0.6890
7.79	6.85	7.32	895.	546.	729.	968.	0.7519	489.	227.	303.	487.5	0.6219
6.85	5.97	6.41	841.	506.	680.	906.	0.7512	416.	185.	289.	455.2	0.6349
5.97	4.95	5.46	793.	489.	631.	836.	0.7547	362.	172.	260.	421.	0.6173
4.95	4.21	4.58	713.	372.	558.	765.	0.7299	345.	153.	240.	385.	0.6242
4.21	4.12	4.17	678.	393.	538.	731.	0.7353	345.	154.	225.	368.	0.6116
4.12	2.88	3.48	647.	382.	513.	671.	0.7634	307.	154.	209.	338.2	0.6179
2.88	2.82	2.83	525.	344.	450.	607.	0.7407	256.	127.	191.	308.	0.6305
2.82	2.05	2.44	546.	321.	420.	557.	0.7535	252.	119.	168.	280.5	0.5988

By Barnaby and Hayden - Average velocities are of 100 observations.

2.49	2.06	2.28	555.	278.	442.	539.5	0.8197	238.	89.	166.7	272.	0.6135
2.06	1.63	1.85	555.	227.	370.	486.	0.7622	227.	71.	146.6	244.5	0.5988
1.63	1.46	1.55	417.	185.	330.5	445.	0.7435	192.	68.	126.6	224.8	0.5634
1.46	1.27	1.37	384.	167.	295.1	418.	0.7067	173.	82.	118.4	218.2	0.5435
1.27	1.10	1.19	332.	179.	270.1	390.	0.6935	156.	69.	105.6	210.5	0.5082
1.10	0.97	1.04	332.	161.	252.5	364.	0.6944	147.	64.	94.5	203.5	0.4604
0.97	0.84	0.91	312.	147.	227.5	341.	0.6667	119.	43.	84.1	171.7	0.4902
0.84	0.68	0.76	278.	109.	207.8	311.5	0.6667	114.	40.	76.7	157.	0.4890
0.68	0.57	0.63	263.	139.	192.8	283.5	0.6788	89.	36.	67.2	146.	0.4608
0.57	0.45	0.51	238.	82.	160.4	255.	0.6329	78.	32.	52.7	129.	0.4090
0.45	0.36	0.41	179.	75.	126.1	229.	0.5495	61.	27.	41.2	115.2	0.3571
0.36	0.28	0.32	152.	55.	103.1	202.	0.5102	47.	15.	31.9	101.7	0.3135

TABLE XI.—*Velocity of Settling of Particles of Galena and Quartz, under Free Settling Conditions.*

By Bardwell, elutriation test—Average diameters are of 10 microscopic fields.

Galena Diam., mm.			Vel'c'ty	Compt Vel'c'y V.	Ratio Average divided by Comptd f.	Quartz Diameter, mm.			Vel'c'y	Compt Vel'c'y V.	Ratio Average divided by Compt f.
Largest	Smallest.	Average.	mm. per Second.	mm. Sec.		Largest.	Smallest.	Average.	mm. per Sec'nd	mm. Sec.	
0.480	0.138	0.345	125.	209.5	0.5970	0.509	0.243	0.369	41.67	109.5	0.3802
0.426	0.119	0.279	111.1	189.0	0.5882	0.413	0.163	0.305	34.48	99.5	0.4202
0.409	0.0764	0.215	88.5	165.8	0.5844	0.389	0.144	0.234	28.57	87.1	0.3279
0.321	0.0523	0.160	74.6	145.5	0.5128	0.321	0.130	0.199	24.39	80.4	0.3080
0.254	0.0445	0.127	62.9	127.2	0.4717	0.298	0.103	0.182	20.41	77.0	0.2658
0.215	0.0296	0.106	52.6	116.2	0.4717	0.254	0.0827	0.156	17.24	71.1	0.2427
0.201	0.0273	0.0967	48.5	111.0	0.3922	0.252	0.0655	0.135	14.49	66.1	0.2193
0.187	0.0242	0.0839	37.0	108.5	0.3571	0.249	0.0808	0.126	12.05	64.0	0.1883
0.165	0.0179	0.0798	31.3	100.1	0.3125	0.237	0.0679	0.121	10.20	62.7	0.1626
0.184	0.0157	0.0714	26.3	95.2	0.2762	0.219	0.0655	0.116	8.55	61.3	0.1395
0.115	0.0195	0.0667	22.2	92.3	0.2404	0.235	0.0569	0.112	7.14	60.3	0.1183
0.0879	0.0125	0.0599	18.5	87.9	0.2105	0.168	0.0608	0.0912	6.02	54.4	0.1111
0.0946	0.00884	0.0572	15.6	85.6	0.1821	0.158	0.0429	0.0846	5.05	52.4	0.0904
0.0853	0.00780	0.0585	13.0	82.6	0.1575	0.147	0.0414	0.0800	4.26	50.9	0.0840
0.0827	0.00832	0.0484	11.0	78.4	0.1408	0.120	0.0364	0.0747	3.57	49.3	0.0725
0.0684	0.00624	0.0487	9.26	74.6	0.1241	0.0972	0.0387	0.0689	3.00	47.3	0.0635
0.0764	0.00676	0.0419	7.75	73.1	0.1059	0.0970	0.0315	0.0629	2.52	45.1	0.0558
0.0741	0.00546	0.0401	6.49	71.6	0.0906	0.0918	0.0317	0.0555	2.12	42.4	0.0500
0.0653	0.00520	0.0390	5.46	70.6	0.0774	0.0842	0.0200	0.0508	1.78	40.4	0.0441
0.0552	0.00419	0.0354	4.59	67.2	0.0683	0.0718	0.0164	0.0478	1.50	39.4	0.0380
0.0499	.....	0.0311	3.86	62.9	0.0614	0.0715	0.0138	0.0425	1.26	37.1	0.0340
0.0440	.....	0.0284	3.25	60.2	0.0541	0.0660	0.0109	0.0377	1.06	35.0	0.0303
0.0431	.....	0.0259	2.72	57.6	0.0473	0.0629	0.0109	0.0344	0.887	33.4	0.0265
0.0419	.....	0.0235	2.29	55.4	0.0413	0.0611	0.00806	0.0319	0.746	32.1	0.0230
0.0408	.....	0.0219	1.92	53.0	0.0362	0.0530	0.00962	0.0282	0.627	30.25	0.0207
0.0394	.....	0.0205	1.61	51.7	0.0312	0.0543	0.00780	0.0267	0.526	29.45	0.0179
0.0301	.....	0.0193	1.36	49.6	0.0274	0.0462	0.00868	0.0253	0.442	28.6	0.0155
0.0267	.....	0.0176	1.14	47.4	0.0241	0.0462	0.00666	0.0232	0.372	27.4	0.0136
0.0305	.....	0.0160	0.959	45.2	0.0212	0.0394	0.00419	0.0209	0.313	26.05	0.0120
0.0233	.....	0.0149	0.806	43.6	0.0185	0.0408	0.00826	0.0188	0.262	24.7	0.0106
0.0198	.....	0.0141	0.676	42.4	0.0159	0.0323	.....	0.0182	0.220	24.3	0.00905
0.0230	.....	0.0134	0.568	41.4	0.0137	0.0301	.....	0.0161	0.185	22.85	0.00810
0.0226	.....	0.0128	0.478	40.5	0.0118	0.0412	.....	0.0144	0.156	21.6	0.00723
0.0173	.....	0.0112	0.401	37.8	0.0106	0.0403	.....	0.0126	0.131	20.1	0.00709
0.0167	.....	0.00956	0.337	34.95	0.00965	0.0318	.....	0.00884	0.110	16.91	0.00649
0.0166	.....	0.00790	0.283	31.8	0.00890	0.0264	.....	0.00589	0.0924	13.81	0.00668
0.0130	.....	0.00700	0.238	29.9	0.00797	.....	.....	.....	.....	.....	.....
0.0124	.....	0.00598	0.200	27.5	0.00727	.....	.....	.....	.....	.....	.....
0.0104	.....	0.00493	0.168	25.1	0.00670	.....	.....	.....	.....	.....	.....
0.0117	.....	0.00131	0.141	23.42	0.00597	.....	.....	.....	.....	.....	.....
0.0112	.....	0.00330	0.118	20.52	0.00575	.....	.....	.....	.....	.....	.....
0.00745	.....	0.00152	0.0995	18.91	0.00715	.....	.....	.....	.....	.....	.....

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Then D is the width of the cube of mineral in meters.

δ is specific gravity of the mineral, 2.65 for quartz, 7.5 for galena.

h is the column of water to balance the grain.

$h = D(\delta - 1)$.

Rittinger assumes that because a column h meters high balances the weight of a stationary grain, therefore the velocity due to h, if rising, is able to prevent the grain from falling; or, in other words, it is the velocity of the fall of the grain. On this basis he tells us,[2] from the formula $V = \sqrt{2gh}$ when $g = 9.8024$ m., $\sqrt{2g} = 4.42773$ m., that the velocity V of settling in water of grains of minerals is

$$V = C\sqrt{D(\delta - 1)}$$

where C is a constant.

C = 2.44 for average grains,
2.73 for roundish grains,
2.37 for long grains,
1.92 for flat grains.

Rittinger's C seems to be made up of $f\sqrt{2g}$ where f is a factor due to friction.

In Table XI. the column marked "Computed velocity" is obtained from Rittinger's formula $V = \sqrt{2g}\sqrt{D(\delta - 1)}$, omitting the f. In the column marked "Ratio of computed divided by average," we have a value for f. The value of this factor is practically constant for grains larger than 1.55 mm. in diameter. For galena it is 0.7558; for quartz, 0.6157. But for grains smaller than 1.55 mm. in diameter the value of f decreases in a most extraordinary degree. This discrepancy between the values shows that Rittinger's universal formula for all minerals is not universal, and that it needs some added factor which will provide for the differences in specific gravity. This may be overcome for practical purposes by simply determining the factors for different specific gravities, as has been done above for quartz and galena.

Mr. G. W. Eastman has kindly made a study of this question, and his line of argument with his conclusions is here given.

In Fig. 6, two curves for quartz and galena were drawn from the average diameters and velocities by using logarithms

[2] *Lehrbuch der Aufbereitungskunde*, p. 191 (1867).

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of the numbers instead of the numbers themselves. The abscissas are the logarithms of D, the ordinates are the logarithms of V. The advantage of the logarithmic curve is its compactness and the ease with which the formulas can be derived from it. A curve made from natural numbers would be many feet long.

The curves show at once two things: that they are practically parallel, and that they are divided in the main into two parts.

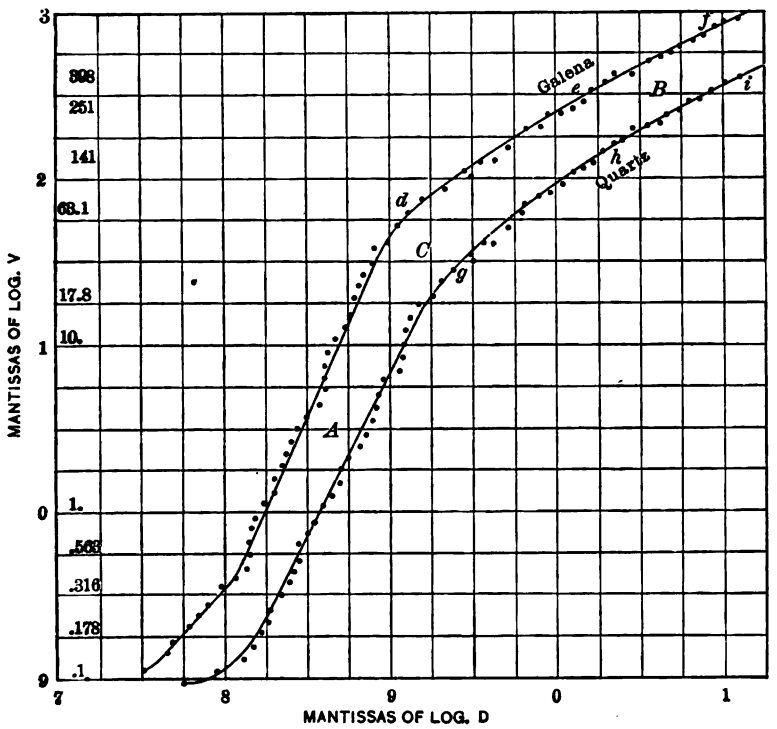

FIG. 6.—LOGARITHMIC CURVES OF AVERAGE VELOCITIES AND DIAMETERS OF THE COMPLETE SERIES.

The points for the smaller grains, which follow one law (the Law of Viscous Resistance), are on a straight line, *A*. The points for the larger grains, which follow another law (the Law of Eddying Resistance), are also on a nearly straight line, *B*. Between the two lines, *A* and *B*, is the critical or transition space, *C*.

The derivation of the Law of Viscous Resistance is as follows:

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For the case of a small sphere falling slowly through a viscous fluid, Sir G. G. Stokes[3] has deduced from purely theoretical considerations for the terminal velocity, V, of the sphere a formula, which to suit the present conditions has been modified as follows:

$$V = \tfrac{2}{9}\, g \left(\frac{\delta - \delta'}{n}\right) r^2$$

where r = radius of the sphere,
g = acceleration due to gravity,
δ = density of the sphere,
δ' = density of the fluid,
n = the co-efficient of viscosity or "inner friction" of the fluid,

the quantities all being expressed in c-g-s. (centimeter-gram-second) units.

For water at 20° C. $\delta' = 1$, and $n = 0.010$; hence the formula would become

$$V = K\,(\delta - 1)\,D^2, \qquad (A)$$

where the constant K should theoretically be the same (about 550) for particles of all densities; but since it involves n, it would change about 2 per cent. per degree for temperatures different from 20° C.

If V and D are taken in millimeters instead of V and r in centimeters, the formula becomes, by substituting known quantities,

$$\left(\frac{V}{10}\right) = \frac{2}{9}\ 0.981 \left(\frac{\delta - 1}{0.01}\right) \left(\frac{D}{2 \times 10}\right)^2,$$

or $V = 545\,(\delta - 1)\,D^2$.

For a given substance $(\delta - 1)$ would also be constant; so we should expect to find that the velocity of settling would be simply a constant times the square of the diameter of the particle, or

$$V = K'\,D^2, \text{ where } K' = K\,(\delta - 1),$$

or taking logs., we have

$$\log. V = \log. K' + 2 \log. D.$$

That such a simple law is followed closely by both quartz and galena is shown very clearly by reference to the logarith-

[3] *Mathematical and Physical Papers*, vol. iii., p. 60 (1901).

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mic plots, Fig. 6. The points on the lines *A* lie very strikingly on a straight line, whose slope is nearly 2 : 1; that is, log. V is increasing just twice as rapidly as log. D, as it should, according to the formula above. The value of log. K′ is the intercept on the V axis (that is, the value of log. V when log. D = 0), from which we get readily the values for K′—namely, 700 for quartz and 4100 for galena. The corresponding values for K in formula (A) would be

$$K = \frac{700}{(2.65 - 1)} = 424 \text{ for quartz,}$$

$$\text{and } K = \frac{4100}{(7.5 - 1)} = 631 \text{ for galena,}$$

values which differ considerably, it is true, but which lie on either side of the theoretical value 550.

The derivation of the Law of Eddying Resistance is as follows: Stokes's equation is derived on the assumption of small velocity and a resistance due entirely to viscosity proper, and it is known not to hold above a certain "critical velocity" when the resistance due to eddying motion set up in the fluid becomes appreciable and important. For such high velocities a complete theory seems to be almost impossible; but Sir Isaac Newton pointed out that the resistance might be expected to vary as the square of the velocity. In other words, $R = kV^2$ where R is the resistance to motion, and k is a constant. Evidently when dynamic equilibrium is attained R is just equal to the effective weight of the particle in the liquid. The effective weight has been shown above to be $D(\delta - 1)$. Substituting this value $D(\delta - 1)$ for R in the equation above, we have

$$D(\delta - 1) = kV^2,$$

from which we get, by extracting the square root, Rittinger's formula:

$$V = C\sqrt{D(\delta - 1)} \qquad (B)$$

(using C outside the radical in place of $\frac{1}{k}$ inside).

We should expect this to hold only when Newton's law of

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resistance is followed, and the results indicate that this is more nearly true, the greater the velocity; that is, when the true viscous resistance plays a continuously less important part, and the eddying resistance an increasingly important part.

The existence of the "critical velocity" and the transition from the Law of Viscous Resistance (A) to the very different Law of Eddying Resistance (B) is strikingly shown on the plot by the decided change in the slope of the lines for both quartz and galena between the lines *A* and *B*. Actually the slope for both quartz and galena does become about $\frac{1}{2}$ from *e* to *f* and from *h* to *i*. The formulas thus indicated are: for quartz, $V = 113 \sqrt{D}$, and for galena, $V = 250 \sqrt{D}$. (The data would be represented over a somewhat larger range, *g* to *i* and *d* to *f*, by the formulas $V = 89\ D^{0.67}$, for quartz; $V = 240\ D^{0.75}$, for galena.)

Bringing in the specific gravities (in other words, finding C for the Rittinger formula), would change these two expressions into $V = 87 \sqrt{(\delta - 1)D}$, for quartz, and $V = 100 \sqrt{(\delta - 1)D}$, for galena. (The constants 87 and 100 would correspond to 2.7 and 3.2 in Rittinger's formula, when V and D are expressed in meters instead of in millimeters.)

We see again a distinct individuality in the constants for the two substances.

The critical velocities are apparently about 63 mm. a second for galena and 28 mm. for quartz, and the corresponding critical diameters are about 0.13 mm. for galena and 0.20 for quartz. Of course, in this neighborhood, neither of the derived formulas will apply very closely. Owing to the decided change here, a simple formula to cover the entire range seems quite out of the question.

It will be noticed that the four or five observations on the smallest galena particles lie a little off the line *A*, as do also two observations on quartz. It is difficult to see why these cases should deviate from Stokes's law, unless the already very slow settling of the particles is made apparently still slower by slight currents in the water, due to temperature changes, which would be almost unavoidable outside of a well-regulated thermostat or chamber supplied with means of maintaining a constant temperature. An empirical formula could be made to fit these few observations, but it seems hardly necessary. In fact,

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the values of the velocity over the whole range can be read off from the plot as accurately and more readily than they could be computed from the formulas.

One interested in the theory of the motion of a solid through a liquid will find a brief summary in Poynting and Thomson's work,[4] and a more complete discussion, including some interesting work on the rate of rising of bubbles and spheres lighter than water, by H. S. Allen.[5] Both references have been freely used in preparing the above.

VI. Summary.

The above discussion of the experiments thus indicates that two quite different laws are followed by settling particles, depending on whether the velocity is above or below a certain transition or critical point. Below this critical velocity the law is expressed by the formula:

$$V = K(\delta - 1)D^2 \qquad (A)$$

and above this critical velocity the law is expressed by the formula of Rittinger,

$$V = C\sqrt{D(\delta - 1)}. \qquad (B)$$

In the preceding formulas, V represents velocity in millimeters per second, E is the specific gravity of the grain, D is its diameter in millimeters, C and K are constants.

The values of K indicated by our experiments are 424 for quartz and 631 for galena; and the values for C are 87 for quartz and 100 for galena.

I believe that in this paper the mineral-dresser will find aid to a solution of his problems when size of grain and velocity of fall are under consideration. For, if he has other minerals than quartz and galena to deal with, he can assume that the velocity of grains of those minerals will be proportional to their specific gravity. This is approximately true, and nearly enough so for all mill purposes.

I wish to give especial credit to Mr. Warren for his good work in planning and executing his part of this paper.

[4] *Text-Book of Physics*, vol. i. (Properties of Matter), p. 221 (1902).

[5] *Philosophical Magazine* (*London, Edinburgh and Dublin Philosophical Magazine and Journal of Science*), Fifth Series, vol. l., pp. 323 to 338, 519 to 534 (1900).

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Chlorination of Gold-Ores; Laboratory-Tests.

BY A. L. SWEETSER, CANANEA, SONORA, MEXICO.

(New York Meeting, April, 1907.)

I. INTRODUCTION.

EXPERIENCE has shown how difficult it is to obtain information regarding laboratory-tests in connection with the chlorination-process for the extraction of gold from its ores, and I therefore present the following method, somewhat in detail, for the benefit of those who may desire to pursue research work in this field.

The ore chosen was a partly-decomposed porphyry, extremely siliceous and comparatively soft, and of an average value per ton of from $25 to $26 in gold and $0.68 in silver. The chemical analysis gave: Fe_2O_3, 3.35; FeS, 1.5; MnO_2, 0.75; $ZnCO_3$, 4; Al_2O_3, 3.20; H_2O, 0.7; SiO_2 and insol., 83.50 per cent.

The ore, first crushed to pass a 5-mesh screen, was coned and quartered until a 45-lb. sample was obtained. Samples for assay and analysis, taken from this lot by means of riffles, were crushed to pass a 100-mesh screen. The original sample was divided into 10-lb. lots, of which one was crushed to pass a 10-mesh screen; another through a 12-; the third through a 20-, and the last through a 30-mesh screen.

Because of the production of an excessive quantity of fine material it was deemed inexpedient to crush finer than 30-mesh size—a decision the wisdom of which is proved by the data in Table I., showing the increasing ratio of slimes, in proportion as the ore is crushed finer.

In order to determine the percentage of fines in the sizes of mesh selected, a weighed sample of ore was taken from each mesh, and a sample from the 10-mesh lot was passed successively through 20-, 40- and 100-mesh screens, and the percentage remaining on each screen determined. This process was repeated with the 12-, 20- and 30-mesh lots, with the results given in Table I.

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TABLE I.—*Screen-Analysis of Crushed Sample.*

Size of Mesh.	Held on 20.	Held on 40.	Held on 100.	Passed 100.
	Per Cent.	Per Cent.	Per Cent.	Per Cent.
10	63	13	11	13
12	45	23	16	16
20	.....	53	24	23
30	.....	32	33	35

II. ROASTING.

When the analysis was complete, some of the ore was chlorinated in a manner to be explained later, and it was found that no practically important extraction could be obtained, due to the fact that the sulphur absorbed a large part of the chlorine and that the gold was not in a suitable form to be attacked by chlorine. In order to reduce the sulphur to a working-limit (about 0.2 per cent.), it was necessary to roast.

For preliminary roasting, a small amount of ore in scorifiers was put in a muffle and the heat gradually raised, the total time of roasting being 2, 3 and 4 hr. Table II. shows that the ore was roasted within the 0.2 per cent. limit of sulphur at the end of 3 hr., and dead roasted at the end of 4 hr.

TABLE II.—*Elimination of Sulphur by Roasting.*

Size of Mesh.	2 Hr.	3 Hr.	4 Hr.
	Per Cent.	Per Cent.	
10	0.13	0.08	trace
12	0.24	0.18	trace
20	0.31	0.17	trace
30	0.26	0.19	trace

The 3-hr. roast was taken as the one on which to work, and in roasting the ore for the chlorination-tests, roasting-dishes were placed in a muffle, the door of which was kept open, the heat being gradually raised to redness; the endeavor being to maintain as great a heat as possible without sintering the ore. A number of results obtained from the 3-hr. roasts were not altogether satisfactory, and the time of roasting was lengthened to 4 hr., which practically corresponds to a dead roast.

III. CHLORINATION.

The chlorination proper was carried out in the following manner in soda-water bottles: The chloride of lime was first

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weighed and charged. Then the 100 g. of ore was introduced, and water in sufficient quantity to give the ore the consistency of mud. Finally, the sulphuric acid was charged and the bottle sealed by pulling the patent stopper in place, inserting a wooden wedge to tighten it, and filling the neck with melted paraffine. It was necessary to conduct this operation with care and rapidity, and to avoid all agitation, because the chlorine soon began to bubble up through the paraffine if the latter did not solidify immediately. It is advisable to have the bottles perfectly dry before charging the chloride of lime, so as to avoid the premature generation of chlorine.

The bottles were placed in an agitator, and rotated for the required time. This agitator consists essentially of a circular disk 2 in. in thickness and 12 in. in diameter. The bottles were secured by rubber bands in six semicircular notches around the periphery. The disk lies in a plane inclined to its axis of rotation at an angle of 80°, and is connected by belt to a $\frac{1}{10}$-h.p. motor, which furnished ample power; 30 rev. per min. was the average speed at which the tests were made.

In the first series of experiments, on unroasted ore, with 2 g. of chloride of lime and 4 g. of sulphuric acid, the samples were agitated for 1 and 2 hr. by hand. In the second and all subsequent tests the agitator was used. The second series was on ore roasted for 3 hr., with 2 g. of chloride of lime and 4 g. of sulphuric acid, the agitation lasting 1, 2, 3, 4 and 5 hr., respectively. These gave results which, though a great improvement over those on unroasted ore, were not entirely satisfactory.

IV. Leaching.

In each case, as soon as the time of chlorination was complete, the bottles were opened and the excess chlorine allowed to escape; then the ore was leached or filtered in sand-filters constructed as follows: A common acid-bottle, upside down, with the bottom removed and a stopper with a pinch-cock inserted in the neck, formed the percolator. Pure quartz was crushed and screened to pass 10-, 20- and 30-mesh screens, respectively. A few pebbles larger than 10-mesh size were placed in the bottom, above which was a layer 1 in. thick of quartz of 10-mesh size, then 0.75 in. of 20-mesh, and lastly 0.5 in. of 30-mesh size, and above the uppermost layer was placed a cloth, to

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catch the ore, and thus preserve the tailings. Too much fine sand prolonged the term of filtration excessively, while too much coarse material allowed the fines of the ore to pass through the filter. It was found impossible to recover all the fines in the tailings for assay-purposes, since a portion passed through the cloth. In some cases a portion of the fines was caught in the sand, while in others the fines passed into the filtrate. It was found that to remove all traces of gold chloride from the tailings each 100-g. charge required about 700 cc. of wash-water, which is equivalent to about 14,000 lb. of water per ton of ore. In practice this amount of water would be considered excessive, and would increase the number of settling-tanks and precipitating-vats to such an extent as to increase seriously the working-cost. The small sand percolator acts like the paper filter used in chemical work, the water being drawn through the cloth along the glass, without coming in contact with the ore, while in the barrel the water must pass through the whole body of ore before reaching the filter. In the mill a further economy of wash-water is often effected by using the last washing of one barrel for the first washing of the next barrel. The washing is considered complete when, on passing hydrogen sulphide through the wash-water, no gold sulphide is precipitated.

V. Treatment of Tailings.

As soon as the leaching is complete, the cloth containing the tailings is removed and the tailings dried, bucked to pass an 80-mesh sieve and assayed. The difference from the original assay is the amount extracted by the chlorine, and may be calculated in terms of per cent. as follows:

Gold before treatment, ...	1.27 oz. per ton.
Gold after treatment, ...	0.96 oz. per ton.
	0.31

$$1.27 : 100 :: 0.31 : x = 24.2 \text{ per cent., etc.}$$

The actual extraction was slightly higher than that indicated by the results obtained in the experiments on account of the inaccuracy of the tailings-assay, which arose from the passage of some of the fines (obviously thoroughly leached) through the cloth, thus giving a higher result for the tailings-assay.

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TABLE III.—*Extraction by Chlorinating Unroasted Ore of a Gold-Value of* $26.25 *Per Ton.*

Size of Mesh.	Time of Chlorination.	Tailings-Assay.	Value of Tailings-Assay.	Percentage of Extraction.
	Hours.	Ounces.		
10	1	0.96	$19.30	24.2
12	1	1.2	24.89	4.8
20	1	1.23	25.42	3.15
30	1	1.27	26.25	0
10	2	0.98	19.60	24.88
12	2	1.27	26.25	0
20	2	1.27	26.25	0
30	2	1.27	26.25	0

NOTE.—2 g. of bleach and 4 g. of sulphuric acid were used.

TABLE IV.—*Extraction by Chlorinating Ore Roasted* 3 *Hours.* (*Value of Original Ore was* $26.25 *Gold Per Ton.*)

Size of Mesh.	Time of Chlorination.	Tailings-Assay.	Value of Tailings-Assay.	Percentage of Extraction.
	Hours.	Ounces.		
10	1	0.13	$2.69	89.75
12	1	0.10	2.07	92.11
10	2	0.20	4.13	84.26
12	2	0.19	3.92	84.06
20	2	0.12	2.48	90.55
30	2	0.12	2.48	90.55
10	3	0.18	3.72	85.82
12	3	0.08	1.66	93.67
20	3	0.16	3.30	87.42
30	3	0.2	4.13	84.26
10	4	0.15	3.10	88.20
12	4	0.1	2.07	92.1
20	4	0.16	3.30	87.42
30	4	0.23	4.75	81.94
10	5	0.96	19.83	24.4
12	5	0.4	8.26	68.52
20	5	0.36	7.18	71.65
30	5	0.32	6.10	76.76

NOTE.—2 g. of bleach and 4 g. of sulphuric acid were used.

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TABLE V.—*Extraction by Chlorinating Ore Roasted 4 Hours. (Value of Original Ore was $26.25 Gold Per Ton.)*

Size of Mesh.	Time of Chlorination.	Tailings-Assay.	Value of Tailings-Assay.	Percentage of Extraction.
	Hours.	Ounces.		
10	1	0.66	$13.64	53
12	1	0.36	7.44	72
20	1	0.05	1.03	96.1
30	1	0.14	3.70	88.9
10	2	0.21	4.34	84.3
12	2	0.11	2.27	91.3
20	2	0.03	0.62	97.6
30	2	0.06	1.24	95.3
10	3	0.45	9.30	64.3
12	3	0.14	3.70	88.2
20	3	0.1	2.07	92.1
30	3	0.08	1.65	92.9
10	4	0.17	3.51	86.6
12	4	0.28	5.78	77.8
20	4	0.02	0.41	98.4
30	4	0.04	0.82	96.1
10	5	0.18	3.70	86
12	5	0.24	4.96	81.2
20	5	0.07	1.44	94.5
30	5	0.06	1.24	94.9

NOTE.—3 g. of bleach and 6 g. of sulphuric acid were used.

VI. DISCUSSION OF TESTS.

The results of the experiments on ore roasted for 3 hr. were unsatisfactory, since the best extraction was only 93.67 per cent. Assuming that the poor extraction was due to the 0.2 per cent. of sulphur left in the ore, experiments were then made on ore that had been roasted 4 hr., which entirely eliminated the sulphur. At the same time the quantity of bleaching powder was increased from 2 to 3 g., and the quantity of sulphuric acid from 4 to 6 g. Under these conditions the results were entirely satisfactory, the best extraction being 98.4 per cent. The percentage of extraction varied with both the size of the ore and the time of chlorination. The former showed an average extraction as follows: 10-mesh size, 72.84 per cent.; 12-mesh, 82.1 per cent.; 20-mesh, 95.7 per cent.; and 30-mesh, 93.6 per cent. With regard to the time of the experiment, the average extraction was: 1 hr., 77.5 per cent.; 2 hr., 92.125 per cent.; 3 hr., 84.37 per cent.; 4 hr., 89.7 per cent.; and 5 hr., 89.15 per cent. The best single test was with 20-mesh size and 4 hr. chlorination, using 3 g. of bleach and 6 g. of sulphuric acid. Accordingly, this size and time were adopted, leaving only the quantity of bleach and of acid to be determined.

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VII. TESTS FOR QUANTITY OF BLEACH AND OF SULPHURIC ACID.

Using ore of 20-mesh size and chlorinating for 4 hr., the results obtained by varying the quantity of sulphuric acid and of bleach are given in Table VI.

TABLE VI.—*Results Obtained by Varying the Quantity of Bleach and of Acid.*

No. of Test.	Quantity of Bleach.	Quantity of Sulphuric Acid.	Tailings-Assay.	Percentage of Extraction.
	Grams.	Grams.	Oz. Per Ton.	
1	0.5	1	0.05	96.1
2	1.0	2	0.055	95.7
3	1.5	3	0.03	97.6
4	2.0	4	0.15	88.2
5	2.5	5	0.04	96.9
6	3.0	6	0.02	98.4
7	3.5	7	0.025	98.1

The results given in Table VI. show that the first, third and sixth tests are the only ones worthy of consideration. Test No. 1 required 10 lb. of bleach and 20 lb. of sulphuric acid per ton of ore. The cost of 10 lb. of bleach at 1 c. is \$0.10, and of 20 lb. of sulphuric acid at 0.6 c. is \$0.12, giving a total cost of \$0.22. Test No. 3 required 30 lb. of bleach and 60 lb. of acid, which cost \$0.66. No. 6 required 60 lb., and 120 lb., costing \$1.32. The value of the gold extracted in Test No. 1 was: $\$26.25 \times 0.961 = \$25.22$; No. 3, $\$26.25 \times 0.976 = \$25.62$, and No. 6, $\$26.25 \times 0.984 = \$25.83$.

Balancing the increased extraction of gold against the increased cost of chemicals, Test No. 1 was the most economical. In the barrel-test based on Nos. 1 and 3 the latter proved to be the better. The most satisfactory results of these tests show that the ore should be 20-mesh size, the time of the roast 4 hr., the quantity of bleach 30 lb., and the quantity of acid 60 lb., per ton of ore treated.

VIII. TREATMENT OF THE GOLD CHLORIDE SOLUTION.

The gold chloride solution was treated with hydrogen sulphide, which precipitated the gold as gold sulphide, the solution being constantly stirred during the operation in order to facilitate settling. After complete precipitation the solution was allowed to settle for a few hours, and the clear supernatant

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liquid was decanted through a filter; finally, the gold sulphide was washed on this filter, dried, and roasted until all traces of sulphur had disappeared.

The gold was refined in scorifiers with test-lead, borax, and silica, the lead button thus obtained cupelled, and the gold weighed. From this weight the percentage of extraction may be determined as a check on the results obtained by the tailings-assay. This method, however, seems of little practical value because of the intricacy of the manipulation required, and the liability of loss of gold in the roasting, and in the burning of the filter-paper on which it is caught.

IX. Barrel-Test.

In order to approximate more nearly to conditions of actual practice, a chlorination-test was made in a small barrel of 100-lb. capacity, basing the experiment on the data obtained from the laboratory experiments.

An iron barrel 2 ft. long and 1.25 ft. in diameter, inside measurements, lined throughout with lead, and provided with the conventional man-hole and inlet- and outlet-valves, was used. An internal wooden frame-work was provided to support the asbestos filter.

A charge of ore, weighing 72 lb., was crushed to 20-mesh size, and roasted for 4 hr. on the floors of a series of muffles. The barrel, having the filter in place, was filled one-third full with water, to which 2.1 lb. (3 per cent. of the weight of the ore) of sulphuric acid was added. The ore was then introduced, followed by 1.05 lb. of bleach (1.5 per cent. of the weight of the ore). The man-hole and valves were quickly and securely closed, and the barrel was rotated for 4 hr. At the end of this time the rotation was stopped, with the filter underneath; the valves were opened and the leaching began. Atmospheric pressure being insufficient to cause rapid filtering, on account of the fines which formed a cement-like cake over the top, compressed air was used to facilitate the leaching. When the leaching was complete, the barrel was closed, rotated a few times, and then opened and emptied. The tailings-sample was taken, when the leaching was finished, by means of a short iron pipe, which was plunged into the ore at different places. Three check-assays gave an average of 0.083 oz. of gold per ton (or $1.74), which shows an extraction of 93.46 per cent.

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During the leaching, 50 gal. of wash-water was used, which is equivalent to 1,338 gal. per ton of ore. This large excess was due to the care taken to remove the last traces of gold chloride from the ore.

These experiments, both laboratory and barrel, demonstrate the suitability of the ore for treatment by chlorination. The original ore assayed $26.25 in gold and $0.68 in silver per ton. Assuming the extraction obtained in the barrel-test to be 93.46 per cent., there was a loss of value of $1.74 in gold, and all of the silver, making a total loss equal to $2.42 per ton.

The smelter-charge for an ore of this character is about $10 per ton, and neglecting the cost of eliminating the sulphur (which in general is in favor of the chlorination-process), there is a gain in chlorination above smelter-charge of $10, less $2.36, equal to $7.64 per ton.

Ores of the character mentioned above can be chlorinated at a cost of from $2 to $5 per ton, which gives a substantial gain over the cost of treatment in a custom-smelter.

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The Formation and Enrichment of Ore-Bearing Veins.

BY GEORGE J. BANCROFT, DENVER, COLO.

(New York Meeting, April, 1907.)

INTRODUCTION.

IT is unnecessary to repeat here the contents of many valuable contributions to this subject which have appeared in the *Transactions* and in the publications of the U. S. Geological Survey. As a basis for the further suggestions of this paper, the following are the most important:

1. The investigation of J. R. Don,[1] showing the gold of certain Australasian veins to have been deposited by ascending solutions, and not by lateral secretion.

2. The theory of Prof. Posepny,[2] distinguishing the vadose from the deep circulation, and ascribing the origin of certain classes of ore-deposits to ascending solutions of the latter class.

3. The theory of Prof. Van Hise,[3] as to the underground circulation and the primary enrichment of veins thereby.

4. The paper of Prof. J. F. Kemp,[4] showing that ore-deposits are largely the products of "expiring vulcanism." Many of Prof. Kemp's ideas have been widely adopted by mining men.

5. The theory of secondary enrichment, so lucidly expounded by Mr. S. F. Emmons.[5] This theory, dealing with the rearrangement of ore-bodies after primary mineralization, has been generally adopted, and seems to me to have been as completely proved as the nature of the case permits.

There are two facts which I think should be constantly borne in mind in formulating a theory of the genesis of ore-

[1] The Genesis of Certain Auriferous Lodes, *Trans.*, xxvii., 621 (1897).

[2] *Genesis of Ore-Deposits*, pp. 1 to 187, and *Trans.*, xxiii., 197 to 369 (1893).

[3] Some Principles Controlling the Deposition of Ores, *Genesis of Ore-Deposits*, pp. 282 to 432; also, *Trans.*, xxx., 27 to 177 (1900).

[4] The Rôle of the Igneous Rocks in the Formation of Veins, *Genesis of Ore-Deposits*, pp. 681 to 709; also, *Trans.*, xxxi., 169 to 198 (1901).

[5] *Genesis of Ore-Deposits*, p. 462, and *Trans.*, xxx., 206 (1900).

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bodies. The first is that commercially valuable mines are relatively few. Countless veins present all the characteristics of good mines, except the values. I suppose that there are a hundred barren veins for every enriched one, and even the latter are rarely enriched to the extent of more than 20 per cent. of their volume. The stope-map of any old mine, showing the proportion of stopes to barren ground, will very seldom indicate that more than 20 per cent. of the vein has been removed.

The second fact is, that nearly all ore-bodies are found in close association with eruptive rocks. So generally is this fact recognized that an old mining man does not like to spend money on a prospect in a new district where there is no "porphyry" ("porphyry," in the broad miners' sense, meaning any kind of eruptive rock).

In view of these two facts, it seems to me that any theory which does not recognize that only exceptional conditions could produce such exceptional results of enrichment, which does not recognize and explain the relationship between "porphyry" and ore, falls short of the mark.

My own observations as a mining engineer have led me to the following tentative views: (I.) that the majority of mineralized veins are the product of expiring vulcanism; (II.) that most of these veins were primarily mineralized by comparatively rich solutions in comparatively short periods of time; (III.) that the solutions derived their metal-values from a comparatively rich source; (IV.) that there is a barysphere containing large amounts of the useful metals; (V.) that eruptions spring from various depths and bring various kinds of magma towards the surface; and (VI.) that only those eruptions which disturb the barysphere, and bring a magma rich in metals sufficiently near the surface to be leached by vein-making solutions, are productive of valuable ore-deposits, other eruptions producing barren veins.

Ore-bodies due to magmatic segregation are not included in this general survey.

These propositions will be successively considered in the present paper.

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I. The Majority of Mineralized Veins Are the Products of Expiring Vulcanism.

This proposition has been so fully demonstrated by Prof. Kemp in his very valuable article entitled The Rôle of the Igneous Rocks in the Formation of Veins,[6] that I feel it would be superfluous to add much to it.

There is, however, one small matter in which I differ with Prof. Kemp—namely, I do not think there is good reason to believe that the surface-water does not sink down into the rocks to very considerable depths. Prof. Kemp notes that many mines are dry to the point of being "dusty," at depths below 1,000 ft. This is true; but in a drift which is being rapidly driven the freshly-broken breast will always be found to be damp. The reason the lower levels of so many mines are dry and dusty is that the evaporation, slow as it is, is nevertheless faster than the very torpid movement of the ground-water. This torpidity of the ground-water in depth is caused by the "tightness" of the rocks. It is a familiar experience in mining that the ground gets tighter the deeper one goes. Of course, in individual districts there may be other obstacles, such as sills of impervious rocks or clay, preventing the ground-water from sinking into the lithosphere; but, in almost every case known to me, the tightness of the rocks in the lower levels will account for any observed diminution of water-flow. Where recent fissuring has occurred, there is generally no diminution of water-flow with depth. Indeed there may be, as at Cripple Creek, an increase.

It is often overlooked, in discussing the saturation of the ground, that in many mining districts the yearly evaporation is greater than the yearly rain-fall less the run-off. This question has been very extensively studied by the U. S. Hydrographic Survey. One of its publications, entitled The Relation of Rain-fall to Run-off, sets forth data on this subject gathered in many places. No general conclusions are drawn; but I figure that in Colorado the run-off is about one-third the rain-fall. The rain-fall varies from 12 to 24 in., or, after deducting the run-off, we have from 8 to 16 in. per year. The capacity for evaporation is more than 3 ft. per year; so that, except in those channels

[6] *Genesis of Ore-Deposits*, pp. 681 to 709, and *Trans.*, xxxi., 189 to 223 (1901).

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where water quickly gathers after a rain, the country must be in a state of continual thirst, and only such water will remain under ground as is held there by capillary attraction. As depth is gained and the water-gathering channels become less frequent, we find the rocks to be moist but not always wet. Similar conditions prevail in many mining regions.

Prof. Van Hise has drawn attention to what he calls the zone of flowage, which begins at depths of from 5,000 to 12,000 m. To my mind the zone of flowage is more a matter of time, and less a matter of depth, than he considers it. I think a channel may exist at much greater depth than his limit for a short time, but when a long time is involved, I think the zone of flowage may come very close to the surface.

It seems to me entirely logical to suppose that a channel may remain open as long as it takes a laccolite to cool and yet gradually close till it is tight, except where quartz or other vein-matter has formed in it. A channel that would stay open when the surrounding rocks were quiet would have a strong tendency to close gradually if those rocks were subjected to forces which cause flexure or other movement in the earth's crust. If this supposition be well founded, it will account for those cases where the quartz is in lenticular masses and mere tight cracks in the rocks represent the veins beyond the limit of any lenticles. Such cases simply indicate that the channels have closed since the vein was formed. It is noticeable that such veins are often found in schist, or other rocks that show the effect of movement and pressure. The matter of open channels is further discussed under my third proposition.

In districts which have been fissured quite recently, like Cripple Creek, we find open fissures and much water as far down (about 1,500 ft.) as the deepest shafts have gone. But even in this camp the deep shafts in the granite are nearly dry at the bottom. Whether this is because the granite was fissured less than the eruptive rocks in the first place, or because the granite is more mobile under pressure and has closed in on its fissures, I am not prepared to say. At all events, the shafts in the granite are dry at horizons where shafts in the eruptive rocks are troubled with a great deal of water; and this water has been conclusively proved to be simply rain-water stored in the vast underground reservoir formed by the countless open

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fissures in the eruptive rocks. There are no volcanic springs active in Cripple Creek to-day. Hence, I think Prof. Kemp gives a wrong impression, and one which he probably did not intend, when he says that many mines are dry to the point of being dusty in depth, which were wet near the surface. I believe that the earth is very generally impregnated with moisture; but I quite agree with Prof. Kemp that this "sea of underground water" is utterly inadequate to account for ore-bodies. In most formations, there is practically no movement in this water below the 500-ft. level. In many cases it will not run into a mine fast enough to equal the slow evaporation; and it is beyond conception that such a torpid agent could accomplish anything in the line of vein-making before a fissure would close up, even at moderate depths.

All mining men have met with "swelling ground," and most of them have known of swelling ground that could not be accounted for by the action of the air admitted by the mine-workings. This kind of swelling is, of course, very much slower than that due to the "slacking" of lime or other rocks influenced by the air; but it shows the general tendency of rocks to close up any opening beneath the surface. Yet this tendency is always a function of time, and time is one of the most puzzling factors in any geological discussion.

It is impossible to state in years how long a geological operation lasted. The most that can be done is to compare the duration of one geological operation with that of another. To my mind, most ore-deposits show that the time consumed in their formation was very short, compared, for instance, with the time necessary to carve out the canyon of the Colorado; hence, I think that a fissure that would in time close up tight, might nevertheless stay open during the (geologically) brief interval required to cool a mass of eruptive rocks.

II. Most of These Veins Were Primarily Mineralized by Comparatively Rich Solutions in Comparatively Short Periods.

That considerable mineralization has been effected where the solutions passed through very small channels, leaving those channels very little altered, is evidenced in many places. The so-called "flats" of the Black Hills are cases in point. In the

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Penobscot mine, for instance, several flat deposits of gold-ore in the sedimentary rocks have been extensively worked. The ore occurs in shoots averaging perhaps 45 ft. wide and 4.5 ft. thick. Each ore-shoot has a vertical fissure coming up through the so-called "quartzite" beneath it; and over these fissures the richest ore is found. These fissures are so small as to be easily overlooked. Those that I saw varied from 0.125 to 0.675 in. in thickness. In some places they were open, while in other places, where the width was about the same, they were filled with quartz, a circumstance which indicates that these fissures have not closed up to any great extent.

That such extensive ore-bodies should be mineralized through such small fissures suggests strongly that the solutions were comparatively rich and that they flowed for a comparatively short time. A long-continued flow, I think, would have either enlarged the little fissures or filled them completely full of quartz.

The Cortez mine, Nevada, and the Lisbon Valley copper-fields of Utah are also cases in point. At these places mineralized solutions came up through hard strata of sedimentary rocks and spread out in soft porous strata, mineralizing considerable areas. In both localities the vertical fissures are small, and show little alteration of the wall-rocks. From specimens I have seen which were brought from Cobalt, Ontario, Can., I would say this is another case in point. The specimens show solid sheets of silver between comparatively unaltered walls.

That the lavas issuing from volcanoes contain large quantities of steam is well known. During the early stages of the eruption of Vesuvius, in 1898, I observed a small lava stream on two occasions, about 6 days apart. The stream was about half a mile long, and was moving very slowly. I presume it had taken three weeks to gain this length, yet it was spitting steam continually from every pore throughout its entire length. I was much impressed with two things: the great amount of steam escaping from a stream 15 ft. wide and 4 ft. deep; and the fact that the lower end continued to advance when it had only a very dull red heat. I wondered whether the escaping steam did not account for its mobility in some way. If all magmas have stored in them as great quantities of water as the Vesuvius lavas and if they tend to discharge it when brought into condi-

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tions of lessened pressure we have here a source of water of no small moment. It is hardly conceivable, however, that this source can alone supply all the water used in vein-making in all cases. For instance, if a magma contained as much as 25 per cent. of water, then a spring running 10 miners' inches would in 200 years exhaust a body of the magma 40 ft. thick and 3,600 acres in extent. I think we must agree, therefore, that in some cases the volcanic waters are serviceable principally in starting and maintaining open waterways and establishing a current. In such cases they are probably joined by other waters, and the combined flow accounts for the volume of water which we find issuing from some mineral springs. Thus we find ore-deposits, such as the immense quartz-veins of the San Juan, in Colorado, or the big quartz-lenses of the Homestake, in S. Dakota, and of the Mother Lode, in California, that seem to have been formed by a generous supply of water; and we find other ore-bodies which indicate that very little water ever circulated through the veins, as, for instance, the ore-shoots of Goldfield, Nev., which occupy single cracks or net-works of cracks, made since the big quartz-reefs were formed, and in which angular corners of the walls frequently stick out into the solid masses of sulphide ore. As the ore-bodies are found in the easily altered country-rocks as well as in the quartz-ledges, such angular corners are the more remarkable. If the mineralized solutions had run a long time, these corners would have been rounded off.

At Cripple Creek, likewise, we find areas where the joints and seams of the country-rock are coated with sylvanite, and where there is no other evidence of vein-making agencies. In the veins themselves silicification is very slight. Kalgoorlie, West Australia, where a great deal of enrichment has taken place with very little silicification or other alteration of the country-rock, is another illustration. Such camps are irreconcilable, in my mind, with the theory that veins were formed by very lean solutions acting through long periods of time. Even the big low-grade quartz-veins seem to indicate an agent much more active than is generally recognized. To my mind, the evidence suggests that some ore-bodies were formed by the magmatic waters or vapors alone, while others were formed or rearranged by considerable volumes of water. There is some reason to believe that the quartz of an ore-body is not always a

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full-blooded brother to the mineral contents of that body; but I will not discuss that matter at this time.

In most genetic processes nature is extremely wasteful. In the formation of a sand-bar in the Mississippi river, hundreds of tons of material pass down the river every day; but it is only an occasional grain that lodges on the sand-bar. Or in the growth of mounds around mineral springs, the water that flows over them is all charged with mineral matter but it is only an occasional atom that lodges on the mound. In view of this consideration, it seems to me that the theory that ore-deposits were formed by leaching the extremely lean, eruptive, or other rocks known to us on the surface, involves one of two rather untenable suppositions. Either we must conclude that nature has operated with a degree of accuracy which is almost unattainable in the laboratory; that she has leached absolutely clean the metal-contents of a rock which had the merest trace to start with, and that she has precipitated every bit of the metal so gathered in an ore-body, leaving the solution absolutely barren; or else, in case it is admitted that nature probably operated with her usual prodigality, we must assume that a tremendously large mass has been subjected to leaching action to form a relatively small ore-body. In accounting for a large ore-body, such as those of our leading copper-camps, it is difficult to understand how the leachings from such a great area as this hypothesis necessitates could have been gathered together into one underground channel. We are forced to assume that veins must branch out downwardly like the limbs and twigs of a giant tree inverted. Such conditions are contrary to ordinary observation. Veins not infrequently unite as depth is gained, but very seldom branch out with depth. This again leads to the conclusion that the solutions that formed ore-bodies were not the extremely dilute solutions which would result from leaching lean rocks.

Another reason for believing that ore-bodies were formed from comparatively rich solutions is the well-known difficulty of precipitating the last trace of any metal in solution. Whoever has had to do with a leaching process, such as cyanidation or chlorination, knows how difficult it is to get into solution the last trace of gold in the ore, and to precipitate the last trace of value from the solution. In fact, this is practically impossible;

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tailings carrying from 30 to 60 cents per ton are considered in most cases to indicate good work; and foul solution that contains no more than 20 or 30 cents of value per ton is considered poor enough to throw away. Yet the surface-rocks, considered by some to be the source of the metals, are much leaner than the poorest tailings; therefore, solutions picking up metals from them must be poorer than the solutions we are forced to discard as worthless—in fact, not less dilute than the sea-water, which Don found to contain 0.071 grain of gold per ton of 2,240 lb. Don was unable to precipitate directly from this sea-water any gold at all, although he used the best precipitants under the most favorable conditions. He made his determinations by slowly evaporating several tons of sea-water and assaying the residue.

As to the time occupied in forming veins, it seems to me that most of the work has been done during the period that Prof. Kemp so fittingly calls that of "expiring vulcanism." This is a relatively short period—so short, in fact, that changes in its conditions may be noticed within a human life-time. Hot springs are very generally associated with expiring vulcanism; and nearly all the hot springs that I know of are noticeably drying up. At Steamboat springs, Nevada, at least two borings have had to be made to bring the flow up to its original capacity. Old settlers at Glenwood springs, Colorado, testify that hot water used to issue from a number of minor vents which are now dry, and that the main streams are slowly decreasing in flow. At Aguacaliente, in Sonora, the water is used for irrigation, and the abandoned fields farthest down the gulch bear mute testimony to the gradually decreasing flow. This spring is a fine example of the fact that hot mineral springs have some source other than rain-water. The only range in this dry country that receives any rain-fall to speak is the Sierra Madre, 100 miles east and across the Yaqui valley. The largest cold springs within a radius of 20 miles are only large enough to supply water for domestic use.

In discussing mineral springs, it must be borne in mind that some of them are no doubt of secondary origin. Thus, Trimble springs, Colorado, gives every evidence of having received its heat and mineral contents from the oxidization of a large body of iron pyrites.

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TABLE I.—*Analyses of Eruptive Rocks from the Telluride, La Plata and Port Orford Quadrangles.*

No.	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22	23
SiO_2	64.72	65.70	52.05	47.32	59.79	57.42	55.53	60.44	43.98	47.75	39.42	38.55	54.19	50.14	60.88	56.48	57.43	50.56	52.12	53.06	71.45	70.33	75.32
TiO_2	0.43	0.72	.....	1.50	0.67	0.86	0.95	0.60	1.18	1.22	.....	tr.	0.12	1.42	0.41	0.81	0.66	1.67	1.38	0.42	0.16	0.27	0.16
Al_2O_3	14.18	15.81	17.96	16.92	17.25	18.48	16.78	16.67	13.30	15.19	1.39	1.32	10.66	15.26	17.71	13.81	17.69	14.49	15.21	12.83	14.58	15.74	13.17
Fe_2O_3	1.58	2.54	4.09	6.21	3.60	3.74	4.06	2.31	3.67	5.05	3.42	5.55	0.52	1.19	2.92	1.73	1.59	1.78	1.88	1.20	0.49	1.43	0.27
FeO	0.40	1.62	6.33	5.94	1.59	2.10	3.35	3.09	6.92	4.95	4.29	2.17	2.26	8.75	2.17	3.95	3.48	10.20	8.75	5.10	0.94	0.83	0.98
CaO	2.62	2.56	8.64	8.51	3.77	6.84	6.96	4.22	10.66	9.98	1.10	0.85	10.76	9.84	4.32	6.69	5.72	10.13	3.75	13.71	2.01	3.38	1.48
BaO	0.28	0.12	.....	0.07	0.14	0.15	0.13	0.12	0.06	0.08	.....	.....	0.04	0.03	0.06	tr.	.....	tr.	.....	.....	0.03	0.09	0.28
SrO	0.21	0.03	.....	0.06	0.11	0.08	0.11	0.11	0.05	0.05	.....	.....	0.05	.....	.....	0.02	0.02	.....	.....	.....	.....	.....	0.02
MgO	0.50	1.62	5.03	5.69	1.24	1.71	3.00	2.18	7.03	6.87	39.68	39.06	11.90	7.21	2.21	8.67	2.73	5.90	6.01	7.50	0.30	0.53	0.42
K_2O	1.82	4.62	1.61	2.02	5.05	3.71	3.57	2.71	1.64	2.60	none.	0.05	1.03	0.95	2.68	0.46	0.58	0.38	0.48	0.05	2.55	1.87	2.14
Na_2O	3.88	3.62	2.99	2.70	5.04	4.52	4.31	5.18	2.15	2.39	.....	0.10	1.35	2.76	4.17	5.03	7.19	2.91	4.88	3.56	7.15	4.33	4.77
P_2O_5	0.08	0.33	0.31	0.96	0.35	0.36	4.47	0.29	0.32	0.25	.....	tr.	tr.	0.24	0.16	0.02	0.18	tr.	0.14	tr.	0.09	0.06	0.04
H_2O	9.50	0.59	0.97	1.28	0.58	0.36	0.64	1.43	1.94	2.52	9.89	11.28	5.93	2.45	2.01	2.69	2.29	1.70	4.37	2.32	0.53	1.36	0.91
CO_2	.....	.....	.....	.....	0.72	.....	0.09	0.48	6.46	1.87	.....	0.51	0.06	.....	.....	.....	0.10	?	0.09	0.25	0.08	tr.	0.03
MnO	.....	.....	0.43	.....	0.20	0.09	0.16	0.13	0.22	0.17	.....	0.05	0.11	tr.	tr.	tr.	0.17	0.25	0.19	0.16	tr.	tr.	tr.
S	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	0.03	tr.	0.04	tr.	tr.	0.02	tr.	tr.	tr.	tr.	tr.	.....
SO_3	.....	0.12	.....	0.19	0.04	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....
Cl	.....	0.03	.....	.....	tr.	0.03	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....
MoO	.....	.....	.....	0.08	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....
LiO	.....	.....	.....	.....	tr.	tr.	tr.	tr.	.....	.....	.....	tr.	?	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....
V_2O_3	.....	.....	.....	.....	.....	.....	0.02	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....
Fe_2S_2	.....	.....	.....	.....	.....	.....	0.04	.....	0.54	0.02	.....	.....	.....	.....	.....	.....	.....	0.28	.....	.....	.....	.....	0.09
NiO,CoO	.....	.....	.....	.....	.....	.....	tr.	.....	0.03	.....	.....	0.13	0.03	.....	.....	.....	.....	tr.	0.08	tr.	.....	.....	.....
Cr_2O_3	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	0.58	0.48	0.15	tr.	.....	tr.	.....	.....	.....	0.06	.....	.....	.....
Total	100.20	100.03	100.41	99.95	100.14	100.45	100.17	99.96	100.15	100.96	99.77	100.13	99.16	99.78	99.70	99.96	99.55	100.25	99.18	100.22	100.31	100.22	100.03
Sp. gr	2.80	2.86	2.89	2.95	2.70	2.77	2.79	2.68	2.91	2.80	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....

DESCRIPTION OF THE ROCKS ANALYZED, ALL FROM U. S. GEOLOGICAL SURVEY REPORTS.

Telluride Quadrangle.—No. 1, black vitrophyre from ridge east of pass between Marshall and Virginius basin. No. 2, monzonite from near lake NW. of San Miguel peak. No. 3, gabbro from Stony mountain. No. 4, gabbro-porphyry from pass south of Mount Sneffles.

La Plata Quadrangle. No. 5, porphyry. No. 6, monzonite from Babcock peak. No. 7, monzonite facies of diorite mass. No. 8, diorite porphyry. No. 9, basic dike rock. No. 10, lamprophyre from Snowstorm peak.

Port Orford Quadrangle.—No. 11, serpentine, 12 miles north of mouth of Boulder creek. No. 12, serpentine, Iron mountain crest, near middle. No. 13, gabbro, SE. slope of Panther mountain. No. 14, gabbro, summit of Bald mountain. No. 15, gabbro, Brush creek 1.5 miles SE. Bald mountain. No. 16, gabbro, SE. ¼ sec. 7, T 343, R 142 on cross-road. No. 17, gabbro, left bank Rogue river, 2 miles below the mouth of the Illinois. No. 18, basalt, Cedar creek, 1.5 miles NE. Ophir. No. 19, basalt, near fork of West Bend trail, 2.5 miles south of Johnson creek. No. 20, basalt, Saw-tooth rock. No. 21, dacite-porphyry, 6 miles west of Big Bend of Rogue river. No. 22, dacite-porphyry, head of Boulder creek. No. 23, dacite-porphyry, south slope of Bald mountain.

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TABLE II.—*Analyses of Spring-Waters.*

Parts in one thousand.

No.	1	2	3	4	5	6	7	8	9	10	11	12
Na	0.1042	0.7743	0.3554	0.3040	0.6116	.....	.....	.....	.....	.....	0.0310	0.0020
K	tr.	0.0669	0.0191	0.0094	0.0630	.....	.....	.....	.....	0.1285	0.0872	0.0128
Ca	0.5060	0.0805	0.0367	0.0121	0.0589	.....	.....	.....	.....	1.0739	.....	0.0012
Mg	.....	0.0010	0.0034	0.0004	0.0604	.....	.....	.....	.....	0.2352	.....	.....
Ba	.....	.....	.....	.....	.....	.....	.....	.....	.....	tr.	.....	.....
Sr	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....
Li	tr.	.....	.....	.....	.....	.....	.....	.....	.....	0.0003	.....	.....
Fe	tr.	.....	.....	.....	.....	.....	.....	.....	.....	0.0027	.....	.....
Mn	tr.	.....	.....	.....	.....	.....	.....	.....	.....	0.0009	.....	.....
Cl	tr.	0.9697	0.2896	0.2070	0.2272	0.9524	.....	.....	.....	4.2730	0.0480	.....
Br	.....	.....	.....	.....	.....	.....	.....	.....	.....	0.0025	.....	.....
I	.....	.....	.....	.....	.....	.....	.....	.....	.....	tr.	.....	.....
Fl	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....
CO_2	0.0852	.....	tr.	.....	0.5787	0.1776	0.2624	1.7513	.....	0.1792	0.0383	0.0089
SO_3	0.1614	0.3555	0.3901	0.3492	0.3131	0.1089	.....	.....	.....	2.0318	.....	0.0040
HPO_4	tr.	.....	.....	.....	.....	.....	.....	.....	tr.	0.0004	.....	0.0179
B_4O_7	.....	.....	.....	.....	.....	0.2174	1.8784	2.4043	tr.	tr.	.....	.....
Al_2O_3	tr.	0.0010	.....	.....	0.0018	0.0003	.....	.....	.....	0.0022	.....	0.0012
SiO_2	tr.	0.2788	0.1136	0.1310	0.1220	0.3106	0.0371	0.0418	0.2464	0.0080	.....	0.0317
H	.....	.....	.....	.....	.....	.....	.....	.....	.....	0.0080	.....	.....
Organic	tr.	.....	.....	.....	.....	.....	0.0050	0.0076	.....	.....	.....	.....
O	.....	0.0194	0.0255	0.0080	0.0825	.....	.....	.....	.....	.....	.....	.....
H_2S	0.0500	.....	.....	.....	.....	0.0005	0.0045	0.0007	.....	.....	.....	.....
RHS and S	.....	.....	.....	.....	.....	0.0069	.....	.....	.....	.....	.....	.....
As_2O_5	.....	.....	.....	.....	.....	0.0036	.....	.....	.....	.....	.....	.....
Sb_2O_3	.....	.....	.....	.....	.....	0.0005	.....	.....	.....	.....	.....	.....
P_2O_5	.....	.....	.....	.....	.....	0.0006	.....	.....	.....	.....	.....	.....
HgS	.....	.....	.....	.....	.....	tr.	.....	.....	.....	.....	.....	.....
FeO	.....	.....	.....	.....	.....	0.0002	.....	.....	.....	.....	.....	.....
MgO	.....	.....	.....	.....	.....	0.0005	.....	.....	.....	.....	.....	.....
Na_2O	.....	.....	.....	.....	.....	0.9193	.....	.....	.....	.....	.....	.....
CaO	.....	.....	.....	.....	.....	0.0005	.....	.....	.....	.....	.....	.....
K_2O	.....	.....	.....	.....	.....	0.1246	.....	.....	.....	.....	.....	.....
$FeCO_3$	.....	.....	.....	.....	.....	.....	.....	0.0010	tr.	2.0398	.....	0.0109
NaCl	.....	.....	.....	.....	.....	.....	1.1027	1.0397	13.6226	.....	.....	0.0243
Alk. Carb	.....	.....	.....	.....	.....	.....	2.0075	0.3809	0.4768	.....	0.0154	.....
KSO_4	.....	.....	.....	.....	.....	.....	.....	.....	0.8005	.....	.....	.....
$CaSO_4$	.....	.....	.....	.....	.....	.....	0.0234	.....	1.1548	.....	.....	.....
$NaSO_4$	.....	.....	.....	.....	.....	.....	.....	0.6890	.....	.....	.....	.....
MgCl	.....	.....	.....	.....	.....	.....	.....	.....	0.1637	.....	.....	.....
KCl	.....	.....	.....	.....	.....	.....	0.0470	0.0747	.....	.....	.....	.....
NaBr	.....	.....	.....	.....	.....	.....	.....	.....	0.0704	.....	.....	.....
Total	0.9068	2.5171	1.1834	1.0211	2.0692	2.8194	5.3675	6.3910	16.0352	9.9814	0.1699	0.1149

SPRINGS.

No. 1, Sulphur springs, Los Angeles; *Annual Report U. S. Geological Survey*, p. 195, 1876. No. 2, Hot spring at Hot Spring station, C. P. R. R.; *Chamberlin and Salisbury's Geology*, pp. 224–225. No. 3, Hot springs at the base of the Granite mountains, Nevada; *Chamberlin and Salisbury's Geology*, pp. 224-225. No. 4, Boiling spring at Honey Lake valley, California; *Chamberlin and Salisbury's Geology*, pp. 224–225. No. 5, Warm spring, Mono basin, California; *Bulletin No. 9, U. S. Geological Survey*, p. 27. No. 6, Steamboat springs, Nevada; G. F. Becker, *Geology of the Quicksilver Deposits of the Pacific Slope; Monograph* xiii., *U. S. Geological Survey*, p. 347. No. 7 and No. 8, two different shafts at Sulphur Bank, California; G. F. Becker, *Geology of the Quicksilver Deposits of the Pacific Slope; Monograph* xiii, *U. S. Geological Survey*, p. 259. No. 9, Glenwood springs, Colorado; *Glenwood Springs Hotel Pamphlet*. No. 10, artesian well at Sheboygan, Wisconsin; C. F. Chandler, *American Chemist*, p. 370, 1876. No. 11, the Mississippi river; W. J. Jones, *Report Louisiana State Board of Health*, p. 370, 1882. No. 12, the Sacramento river; W. J. Jones, *Report California State Board of Health*, 1878.

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Table I. contains analyses of 23 eruptive rocks, and Table II. the analyses of 12 spring-waters, the first nine of the latter being hot springs, the tenth an artesian well, and the eleventh and twelfth river-waters. The rock analyses, which are from representative surface eruptives, show that these rocks must be very lean indeed in the useful metals. Mr. Waldemar Lindgren[7] mentions the finding of traces of pyrite, chalcopyrite and galena in the gray gneiss of Freiberg. Prof. Kemp mentions the finding of various metals in various eruptive rocks, but does not give any quantitative analyses.[8]

Don was unable to find any gold at all in the rocks he examined except in association with iron pyrites, which latter gave evidence of being the result of vein-making agencies. Of course, it may be said that all the analyses of the eruptive rocks are made after they have been leached out. If it could be shown that the surface eruptive rocks have a tendency to throw off metals, as they do steam and sulphur, during the cooling process this would remove many of my objections to considering them the source of the metals in our ore-bodies. In the lack of such proof, however, we must recognize that they are extremely lean, and therefore a very unlikely source of mineral wealth.

Of the four rocks from the Telluride quadrangle only one shows manganese; yet rhodonite and rhodocrosite are very common to the veins of this district. Of the six analyses from the La Plata quadrangle, every one shows manganese; yet it has been my observation that manganese is a rare constituent of the veins of this quadrangle. In the 13 analyses of rock from the Port Orford quadrangle a wide range of minerals is seen, which often are found in veins, yet the veins of this locality contain little but quartz, pyrite, chalcopyrite and gold.[9]

The table of spring-waters was rather surprising to me in that it shows that an artesian well-water may contain as much mineral matter in solution as the average hot spring. I do not see that a comparison of the minerals found in hot springs with those minerals found in the eruptive rocks is very instructive. Unfortunately, I have no complete analyses of all the rocks im-

[7] Metasomatic Processes in Fissure-Veins, *Trans.*, xxx., 659 *et seq.* (1900).

[8] The Rôle of the Igneous Rocks in the Formation of Veins, *Genesis of Ore-Deposits*, pp. 681 to 709, and *Trans.*, xxxi., 169 to 198 (1901.)

[9] *Port Orford Folio, U. S. Geological Survey.*

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mediately surrounding a hot spring. If such analyses were available they would be very instructive. As it is, the only distinguishing characteristic of hot springs brought out by the analyses is the presence of sulphur and sulphur gases and of chlorine combinations. Fresh lava, we know, gives off fumes of sulphur and chlorine; hence it is natural to connect hot sulphur springs with fresh eruptives.

The analyses given in Table I. show very few of the useful metals in solution, but Posepny in his *Genesis of Ore-Deposits* mentions that lead occurs in the springs of Rippoldsau (according to Will, 1.6 to 3.7 mg. per ton), and in the Kissingen spring (13 mg. per ton), and quotes from G. Bischof, as follows, the maxima found in mineral springs up to 1854, in milligrams per ton of water:[10] Arsenious acid, 1.5; antimony oxide, 0.1; zinc oxide (sulphate), 13.3; lead oxide, 0.1; copper oxide, 6.4; tin oxide, 0.1.

It is not mentioned whether or not any of these may have been enriched in a secondary way. It would not, however, be surprising if no spring among those analyzed had been primarily enriched with useful metals. We have only a few analyses; ore-bodies are very rare things, and give evidence of having been made in comparatively short periods, while veins are very common things; so that, granted the hot springs are making veins, the probabilities are that they are nearly all making barren veins or barren parts of veins. Posepny found that the Sulphur Bank spring, whose enrichment is probably primary, carried small quantities of mercurial sulphide in suspension. At least he found this material on his filter-paper after filtering the water, but he found only a trace of mercury in the water. Unless we accept this case, I have not read of any in which a mineral spring has been "caught in the act" of making an ore-body, and this only emphasizes my belief that ore-bodies are formed in relatively short periods of time. Ore-bodies seem to have been formed in every geological age since early paleozoic times; and if we grant that each mineralized district was enriched in a relatively short period, it would not be strange if very few ore-bodies, or none at all, were in process of formation at the present moment.

[10] *Genesis of Ore-Deposits*, p. 47.

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We know that many veins are now in process of formation; and we find that many springs carry in solution the materials found in veins, showing that nature is prodigal in her methods. Probably a small part only of the vein-making contents of any spring is deposited in the underground channel; the rest "goes down the creek." If we should find a mineral spring in the process of forming an ore-body, we might expect most of the metallic constituents of the solution to have remained in it after it issued from the ground.

It is true that we analyze spring-waters after they have passed through the zone of precipitation; and it is conceivable that a water showing at the surface no trace of metal may have carried material quantities of metal before it reached the zone of precipitation. But in that case the water must have been completely robbed of metal-values in the zone of precipitation; and it is hard to understand how such a clean precipitation could be effected by such precipitating-agents as we attribute to the superficial zone. It is easier to believe that we have not as yet found an ore-body forming or a spring engaged in forming one.

It has been asserted lately that the curative effect of some mineral springs is largely due to radio-activity. Nearly all the radio-active metals are of high specific gravity; so that the association of mineral springs with ore-bodies or with magmas rich in heavy metals is further indicated in this way.

Glenwood springs, Colorado, and Hot springs, Arkansas, have both been found to be slightly radio-active. But the springs most remarkable in this respect hitherto discovered are the Doughty springs, in Delta county, Colo. Dr. Wm. P. Headden has made some very fine radiographs from the sinter surrounding these springs.[11]

There are several springs, and the analyses differ somewhat; but the principal radio-active spring (called the Drinking spring) has the following analysis, in parts per 1,000: Na, 0.045863; K, 0.001576; Li, 0.000446; NH_4, 0.000068; Ca, 0.005272; Ba, 0.000192; Sr, 0.000150; Mg, 0.003230; Fe, 0.000026; Al, 0.000054; Mn, 0.000060; Zn, trace; Cl, 0.019762; Br, 0.000065; I, trace; SO_4, 0.013022; SiO_2, 0.000696; BO_2, 0.000174; total, 0.090656.

[11] *Proceedings Colorado Scientific Society*, vol. viii., pp. 1 to 30 (1905).

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These are not hot springs; and whether or not they are springs of primary enrichment is not clearly shown. It will be noted from the analysis that Na, Cl, and SO_4 are the principal substances found in solution; but the distinguishing feature of the spring is the barium sulphate which the water is actively precipitating on the mound around the spring. Dr. Headden says: "The deposition of practically pure baric sulphate by a mineral spring is, so far as I have been able to find, a unique fact."[12] The radium is intimately associated with the barium sulphate.

III. The Solutions Derived Their Metal-Values from a Comparatively Rich Source.

This follows necessarily, if it be admitted that ore-bodies give unmistakable evidence that they were formed by rich solutions. If we believe that the source of the values was the surface-rocks, but admit that "expiring vulcanism" set matters in motion for vein-making, we should expect all veins to show a certain amount of concentration of mineral values; at least all of the veins in the vicinity of eruptive rocks. But the complete barrenness of most veins, even in mining districts, is one of the hard facts that are pressed home upon every experienced mining man. There are also, of course, countless absolutely barren veins and dislocations outside the mining districts.

Some writers suppose that the surface eruptive rocks carry appreciably more mineral than other rocks, and that they are the source of the mineral in the ore-deposits. It is, however, a common observation that the characteristic eruptive rocks of a mining camp are not confined to the mineralized area. As examples, I will mention Cripple Creek, Colo., The Homestake, S. D., Kalgoorlie, W. A., Monte Christo, Wash., Goldfield, Nev., Arizona King, Ariz., and El Trinidad, Sonora. It is also common that these rocks carry either no values at all or a metal that is not characteristic of the camp. In the analyses given in Table I. it will be noticed that the eruptives of the La Plata quadrangle all carry manganese, while only one of those in the Telluride quadrangle carries manganese. Rhodocrosite and rhodonite are very prevalent in the Telluride quadrangle, but only occasionally met with in the La Platas, if my own

[12] *Proceedings Colorado Scientific Society*, vol. viii., p. 26 (1905).

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limited observations are to be relied upon. I think that all this points to the exceptional and unusual sources for such ore-bodies.

The principal objection to a deep-seated and rich source for the mineralization of veins which I have read is that of Van Hise, who, calling attention to the limits of the zone of fracture, says:[13]

"On the assumptions (a) that the strength of the rocks is the same as at the surface, (b) that the rocks are all of the same kind, (c) that the temperature is the same as at the surface, (d) that the water present does not make any difference in the character of deformation, (e) that the rocks yield as readily by fracture as by flowage, (f) that the rocks break as readily by fracture when the deformation is slow as when it is rapid, and (g) that the rocks are among the strongest, I have calculated that the maximum depth of the upper part of the zone of flowage under mass-static conditions can not be greater than 12,000 meters."

He concludes that the practical limit in depth of the zone of fracture is about 5,000 m., and that a fissure would close almost at once at a depth of 12,000 meters.

The resistance of a large number of rocks to binding and crushing has been determined; but such figures give us no satisfactory basis for the calculation here involved. It is necessary to consider also the "arching" of any material, even of crushed material. Loose coke and ore have no strength whatever to resist flexure, yet they will "bridge" a blast-furnace, and broken ore will often arch in an ore-chute and choke it up.

If Prof. Van Hise's conclusion is correct, why does not rock-flowage prevent the continued existence of mountain-peaks 5,000 m. high, and of springs at the base of such masses?[14] If a fissure could not exist at a given depth, how can a peak exist to an equal height? Such a peak may represent the foot-wall of a rather flat fissure, the opposite side of which has been removed. Would the absence of the opposite side prevent the action of rock-flowage? Has the phenomenon of rock-flowage ever been observed, bulging out the solid rock at the base of a peak or precipice?

If a spring can flow from under a mass of rock 20,000 ft. high, why could not a fissure exist 20,000 ft. below the surface?

[13] A Treatise on Metamorphism, *Monograph* xlvii., *U. S. Geological Survey*, p. 189 (1904).

[14] According to Mr. M. W. Conway's *Climbing and Exploration in the Karakoram-Himalayas* (vol. i., p. 486, 1894), Peak K2 of that range is 28,000 ft. above sea-level, and many other peaks exceed 20,000 ft. In Bolivia there are mountains rising somewhat abruptly 21,000 or 22,000 ft. from sea-level. Many other instances could be cited, from British Columbia and Alaska.

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Again, the effect of rock-flowage, whatever it may be, is admittedly slow, and must be subject to arrest or diversion by the greater force of rock-movements in mass. Such movements, indicated in innumerable instances by their geological effects, are continually presented for our observation in non-volcanic earthquakes, like those at Charleston, San Francisco, Jamaica, etc., and are reported daily by the seismometers of the world. Is it reasonable to believe that a movement felt at a horizontal distance of 10,000 miles has had no effect below the depth of 5,000 or 12,000 m. (3 or 7 miles)? If it has had such effect, it must have counteracted the previous work of rock-flowage, and opened new fissures, upon which that slow agency must commence operations *de novo*.

Very recent seismometric observations (preceding by a few weeks the Jamaica earthquake of January, 1907) reported a submarine earthquake in the deepest part of the Pacific, far exceeding in intensity and energy anything hitherto observed on land. Such earthquakes occur under, say, 24,000 ft. of sea-water (the maximum depth off the coast of Asia is greater than that), equivalent in weight to 10,000 ft. of rock; yet they not only break the sea-bottom, but possess surplus energy enough to lift the sea itself, producing enormous tidal waves. May we not safely conclude that they might still occur under a greater superincumbent pressure, and, in particular, that 15,000 or 20,000 ft. of rock, the pressure of which, producing rock-flowage, operates much more slowly than an equal pressure producing water-flowage, would not necessarily prevent an earthquake sufficient to make and, for a time at least, maintain fissures?[15]

Again, it is not proved that the pressure which would close an open crack by rock-flowage would be sufficient to change the density of the rock itself, and close all pores and capillary passages in it. On the contrary, it is probable that the slow deformation of rocks under pressure takes place without fracture or change of density. This, at least, is indicated by the early experiments of Sorby and others, and by the actual observed conditions of limestone, etc., which have been thus changed in form, yet show their original structure. It follows, apparently, that the assumed lower limit of rock-flowage is not necessarily the limit of rock permeability, and also that any interruption

[15] This consideration was suggested to me in a private communication by the Secretary of the Institute.

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of the closing of a fissure by rock-flowage might continue for an indefinite period, or until rock-flowage in new directions, or a forced change in the density of the rock, had completed the process thus interrupted. This leaves room for the hypothesis here advanced, which requires, not the endless persistence, but only the existence for a sufficient period, of deep channels of circulation.

Now, there is here no question of an absolutely open fissure, with walls nowhere in contact. On the contrary, fissures are almost always formed by movements of one wall relatively to the other, and are almost always "closed" to a certain extent by the "misfit" contact of the walls in their new relative position. This leaves more or less continuous and connected channels for underground waters and gases, the further closing of which by the pressure of the inclosing rocks may be long delayed by the fact that the pressure tending to close the openings must overcome the resistance of the solid masses which are keeping them open and the "arching" of the material around such openings. It must be admitted that the complete "squeezing out" of all such residual and interstitial cavities is likely to be a much slower process than the closing of altogether open and continuous fissures.

In view of the foregoing considerations, I can see no reason why small open channels may not exist as far below the surface of the earth as mountain peaks extend above the average gravity-level; and, moreover, as these channels would doubtless be filled with water, a practically incompressible liquid, having material weight of its own, I think we may conclude that they would resist closing all the more on that account, and, indeed, could not be completely closed, unless the water were provided with the means of escape—in other words, with channels!

Prof. Van Hise calls attention to the fact that at a certain depth the critical temperature of water (360° C.) would be reached, and that it could not exist as water below that depth. But it does not follow that the form in which it could exist would not possess equal density and solvent power.

According to the rate of increase assumed by many writers—1° C. for every 30 m. of added depth—the critical temperature would be reached at 10,350 m., if 15° C. be taken as the temperature of the surface. But we are scarcely warranted in as-

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suming that rate as uniform to great depths. M. Walferdin, by a series of careful observations in two shafts at Creuzot, proved that down to a depth of 1,800 ft. the increase in temperature amounted to 1° F. for every 55 ft. of descent; but below the depth named the rate of increase was as great as 1° F. for every 44 ft. On the other hand, in the great boring of Grenelle, at Paris, the increase in temperature down to 740 ft. was 1° F. for every 50 ft.; but from 740 to 1,600 ft. it diminished to 1° F. for every 75 ft. A similar remarkable fact was shown in the Sperenberg boring, near Berlin, where the rate of increase for 1,900 ft. was 1° F. for every 55 ft., and for the next 2,000 ft. only 1° F. for every 62 ft. In the deep well at Buda Pesth there was actually found a decline in temperature below the depth of 3,000 ft.

A list of 164 wells, from 400 to 2,220 ft. deep, bored in the United States,[16] shows irregularities of temperature not to be referred to any general formula. To the rule mentioned above—namely, 1° C. for every 30 m. of added depth—there are far more exceptions than confirmations. No doubt these variations are due to local physical or chemical causes; and, in like manner, it must be conceded that under conditions of expiring vulcanism very high temperatures may prevail, probably even beyond the critical temperature of water. But it seems unsafe to reckon upon a transcendently hot interior of the mass of the earth.

Finally, it is not safe to assume that this mass is under such pressure as to be precluded from all movement whatever below a few thousand feet of its 4,000 miles of radius; and a movement causing displacement would give opportunity for the rise of a heavy magma to a higher level.

In view of the above considerations, I find nothing precluding the idea that the solutions which have formed ore-bodies have had comparatively rich sources, and that these sources were very likely laccolites of heavy magmas, brought up from the barysphere into the lower part of the zone of fracture.

IV. There is a Barysphere Containing Large Amounts of the Useful Metals.

This, I think, has never been seriously questioned. Physicists and astronomers have weighed the earth and found it not

[16] *Water Supply and Irrigation Paper* No. 149, *U. S. Geological Survey* (1905).

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"wanting," but over-weight. R. von Sternbeck determined the specific gravity of the earth to be 5.6, while the average density of the surface-rock is 2.5. Chamberlin and Salisbury in their treatise on geology give the specific gravity of the earth as 5.57, and that of the lithosphere as 2.7. There is a theory that the greater relative weight of the earth is caused by pressure alone; that the material is the same throughout, but that the pressure has made the interior rocks more dense. I believe it has been demonstrated that rocks do yield somewhat to such pressure as may be artificially applied; but such evidence comes far short of the proof here required. To satisfy this theory it would be necessary for the rocks to be compressed, near the center of the earth, to one-fourth their volume at the surface. Against this hypothesis, we have the facts that magmas of very different specific gravity issue from the interior of the earth, and that eruptive rocks, as a class, are heavier than surface-rocks.

Van Hise remarks:[17]

"It is noticeable in the altered rocks that in proportion as deep-seated metamorphism is advanced the heavier (of the above) minerals appear."

V. Eruptions Spring from Various Depths and Bring Various Kinds of Magma Toward the Surface.

This seems to me to be shown by what is known of eruptions. As vents filled with molten material would not be subject to the causes limiting the depths of water-channels there is no limit to the depths to which we may expect them to extend. I understand that the majority of both astronomers and geologists regard the earth's interior as solid and rigid as steel. Eruptions are generally considered to be the result of local stress and friction. Just what is the cause of the stress and just how the force is applied are matters of discussion. A very simple explanation, but one which does not seem to appeal to most writers, is that the axis of the earth is gradually shifting, and the earth being an oblate spheroid has to keep rearranging its mass to suit the new positions of the axis.

At Cananea, Sonora, in 1902, I saw an illustration of a volcano on a very small scale. A block of heavy iron gossan, constituting, roughly, a cube of about 200 ft. on a side, or 8,000,-

[17] A Treatise on Metamorphism, *Monograph* xlvii., *U. S. Geological Survey*, p. 183 (1904).

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000 cu. ft. in volume, had been undermined, and slipped down 6 ft., crushing the timbers. The heat produced underground, near the foot-wall, was intense, and on the surface two or three small jets of steam appeared. If a little block of ground like that, slipping 6 ft., could generate sufficient heat to produce such jets of steam, it is easy to understand how the movements of a large region might incidentally produce a volcano or two. Suppose, for example, that an eruption is caused by a force which produces faulting under great pressure. The first effect may be confined to the lithosphere, so that barren magmas are squeezed out. But, as the force gathers intensity, the fault extends into the barysphere, and some of the latter is forced upward, crowding out the lighter lavas above it. By reason of its greater specific gravity it floats the lighter rocks above it, and forms, within the reach of underground waters, a laccolite, which may subsequently become the source of valuable mineral deposits.

Chester Wells Purington, in the Telluride Folio of the U. S. Geological Survey, says, in effect, that the basic parts of eruptive rocks, such as hornblende, augite, biotite, contain more of the useful metals than the other parts, and deems it probable that the mother magma had a basic portion, which might be the source of the metals in the ore-deposits. His idea and mine are not widely at variance.

VI. Only Those Eruptions which Disturb the Barysphere and Bring a Magma Rich in Metals Sufficiently Near the Surface to be Leached by Vein-Making Solutions are Productive of Valuable Ore-Deposits, Other Eruptions Producing Barren Veins.

In support of this proposition there are many indications not mentioned above. In many mining districts there have been successive eruptions, but the ore-bodies are definitely associated with one eruption and appear to have no relationship with the others. Thus, at Butte, Mont., the ore-bodies are associated with a quartz-porphyry eruption, while the acid granite, basic granite and rhyolite eruptions produced no ore-bodies. At Cripple Creek we have a whole series of eruptions; but the mineralization of the veins followed on the heels of the nepheline-basalt eruption.

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We often find in a mining district several series of veins, only one of which bears mineral values. At Rico, Colo., for instance, there is a series, terminating at the so-called "contact," which has been enriched with silver and other metals; and there is another series of strong quartz-veins not thus enriched. These facts suggest that, among the eruptions, one must have been radically different from the others. Yet analyses of the surface-rocks do not reveal any startling differences. It seems evident that the mineralized series of veins must have been formed from a source radically different from that of the barren ones. My explanation is, that in a series of eruptions, one may have been sufficiently deep-seated to disturb the barysphere and force some of its material toward the surface. It would never reach the surface, because its specific gravity would cause it to form, sooner or later, a laccolite, floating the surface-rocks. But, in exceptional cases, it might rise far enough to become subject to the agencies which make mineral-bearing veins.

I presume that the barysphere includes different kinds of unsegregated magmas. It may be built up concentrically, or it may be simply spotted, as the surface is, with different rocks. A laccolite of magma rich in copper might give rise to a surface region yielding copper; one rich in gold might become the origin of a gold-bearing district, etc.

I do not mean that the constituents of the magma would govern entirely. It is conceivable that conditions of the solution and precipitation of the metals might also be influential. But this general hypothesis suggests an explanation of those cases in which totally different kinds of ore-deposits occur in the same surface-rocks, close together, and under conditions apparently similar, except as to age. Butte, Rico and Leadville are cases in point. At Butte there is a great mass of dark, basic granite, which contains two vein-systems. In the southern part of the camp are the famous veins of copper, containing sulphide ore-bodies with more or less quartz. The northern system produces ores of silver, lead, zinc and iron. Prof. Kemp has called attention to the fact that the northern ores are abundantly associated with manganese minerals, especially rhodonite; that no manganese occurs in the copper belt and no copper in the silver belt; and infers that "such results could originate only in different deep-seated sources."

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This hypothesis offers also an explanation of cases in which there is an extensive surface-area, showing similar eruptive rocks throughout, yet only a small part of which has been mineralized. Thus, in areas like southern Nevada and the Yaqui River country of Sonora, there are vast quantities of eruptive rocks of much the same kinds, but only in isolated localities have paying veins been found. Sometimes these localities are, and sometimes they are not, characterized by a trifling exposure of a peculiar eruptive rock. In the former case, the trifling surface manifestation seems utterly inadequate to account for the very exceptional vein-contents of the localities.

Cripple Creek is another case in point. The whole Arkansas plateau is prolific of all the rocks characteristic of Cripple Creek (unless, it may be, the basalt dikes). The largest masses of phonolite I know of are found, as in Grouse mountain and Little Pisgah peak, outside the productive area, while around Saddle mountain and at Bare hills there are large masses of andesitic breccia, yet no ore-deposits. At Globe, Ariz., there is an extensive area, northwest of the camp, that has the same formation as that surrounding the mines; but thus far no ore-deposits of value have been found in it, though it is not lacking in veins.

Of Grass valley, Cal., the U. S. Geological Survey folio says: "The veins occur in almost any one of the many rocks making up the bed-rock series. Excellent mines are located in the grano-diorite, diabase, slate and schists." Evidently the surface-eruptives did not govern in this case.

In the case of several eruptions, only one of which is associated with ore-bodies, the theory would be that the one associated with the ore-bodies was the deep-seated one, which brought some of the mineralized magma within reach of the vein-making agencies, and that, while the surface-manifestation may have been weak, and not different essentially from other eruptions in the same locality, the eruption in depth was radically different.

As to the series of veins in a given district, we would say that the barren ones were the products of the shallow eruptions, while the rich ones were the product of an eruption that brought a rich magma surface-ward. In the case of a large area of eruptive rocks containing a very small mineralized dis-

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trict, it seems to me hard to understand why the mineralization is not much more general, if the surface eruptives are accountable for the metal-values. If, however, these values came from a relatively small buried mass of very richly mineralized eruptive rock, the restricted mineralized surface-area is at once explained.

Again, there are occasional mining districts in which no eruptive rocks at all appear on the surface, such as the zinc-lead deposits of southwestern Illinois, and the Otago gold-fields of the South Island of New Zealand. (The latter are described by Rickard in his discussion of Posepny's *Genesis of Ore-Deposits*.) In such cases the influence of a richly mineralized underlying laccolite is highly reasonable.

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The White Knob Copper-Deposits, Mackay, Idaho.

BY J. F. KEMP, NEW YORK, N. Y., AND C. G. GUNTHER, CLIFTON, ARIZ.

(New York Meeting, April, 1907.)

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I. Introduction.

The White Knob copper-deposits are situated about three miles south of Mackay, on the Salmon River branch of the Oregon Short Line Railroad, in Custer county, Idaho. An outline-map of this district is given in Fig. 1. The deposits have been known and spasmodically worked for many years, and a total of over 3.5 miles of tunnels, shafts and other workings has been driven with a view to their development. These extensive excavations have exposed the deposits sufficiently to permit a careful geological examination.

The peculiar features which give special interest to this paper are the branching, tree-like form of the ore-bodies; the absence of zones of secondary enrichment in the partly oxidized pyritic deposits, and the fact that while the deposits are associated in a general way with the contact of an eruptive rock with limestone, which it penetrates, the garnetization has taken place not in the limestone, as is the usual case, but in the igneous

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rock itself. Contact zones in the limestones practically fail. A new type of ore-body is thus afforded.

The field-work on which this paper is based was done by Mr. C. G. Gunther during a residence of 17 months at the mine. Much assistance and advice was received from Mr. W. L. Austin, who is thoroughly familiar with the mines and to whom the writers are indebted for the suggestion of the method of formation subsequently advocated. The notes, collections and maps have been worked up in the laboratory by Prof. J. F. Kemp.

II. Topographical Features of the Region.

Lost River valley, in which the town of Mackay lies, runs in an easterly direction, and is flanked on either side by ranges of

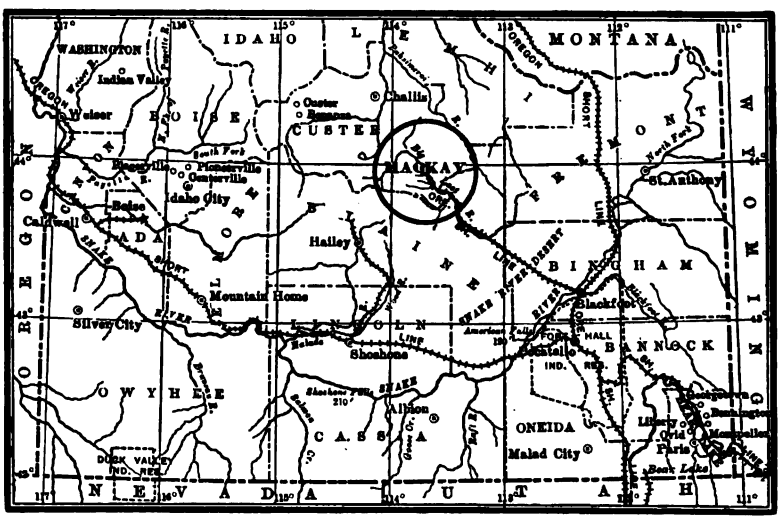

Fig. 1.—Outline-Map of Southern Idaho, Showing the Location of Mackay.

lofty mountains. Lost river itself rises in a spur of the Salmon River mountains and flows easterly, finally disappearing at the mouth of the valley under the lava-beds of the great Snake River flow, as shown in Fig. 1.

The mountains to the north, comprising the Lost River range, consist largely of sedimentary rocks; no discoveries of ores have been reported from them and but little is known of their geological features. The range rising to the south of the valley, and termed locally the "Saw Tooth," appears to consist principally of a core of granitic rocks flanked by limestones.

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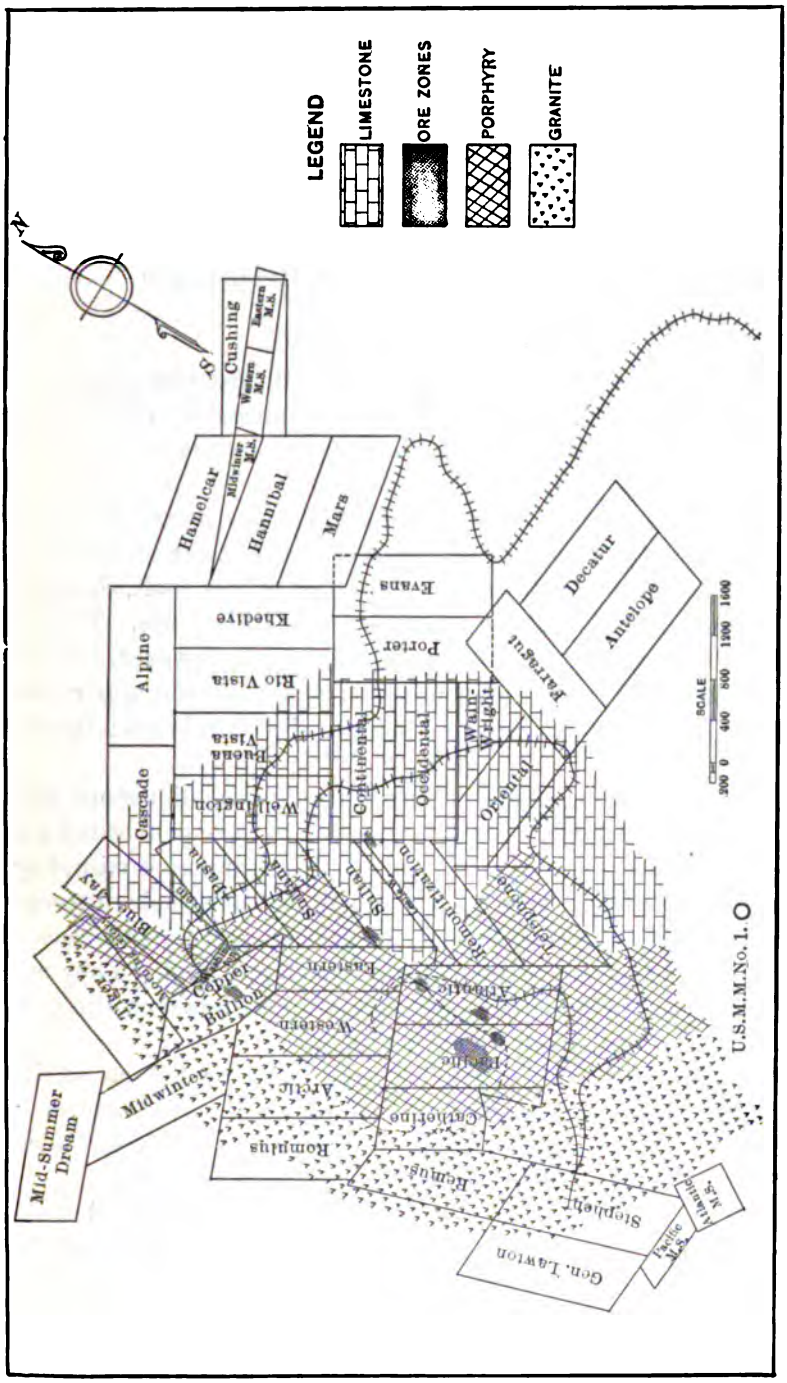

FIG. 2.—SURFACE-MAP OF THE CLAIMS AND GEOLOGICAL FORMATIONS.

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In the vicinity of the mines the topography is varied and accentuated. The granite on the southwest, as shown on the geological map, Fig. 3, forms the highest ground. Between it and the limestone is the belt of ore-bearing porphyry, which has weathered readily and has yielded gentle slopes. The limestone is more rugged and has produced a prominent ridge by its resistance to erosion.

III. Geological Relations of the Mineralized Area.

1. *Structural Features.*

The ore-bearing porphyry varies in width from its maximum of 1,500 ft., in the central portion of its exposure, to a decreasing cross-section at its extremities. In amount it is much less than either the granite or the limestone.

The granite, which on the surface outcrops several hundred feet south of the main shaft, extends on the lower tunnel level for several hundred feet north of the foot of this shaft, showing the fundamental position of the rock at this point. This is well illustrated by the section, Fig. 5, which, however, not being taken on a north and south line, represents the upper contact of the granite as having a slighter dip than is actually the case.

Several dikes of trachyte course through the limestone and the quartz-porphyry, but are not known to have penetrated the granite to any great extent. These dikes are approximately parallel, striking a few degrees north of east; their outcrops are with difficulty traceable on the surface, and for this reason have not been shown on the surface-map, Fig. 2. They are well exposed in the mine, however, and are shown on the geological map of the 700 ft.-level, Fig. 4.

At the point marked "3," Fig. 3, a dike or tongue of the quartz-porphyry extends into the limestone; a shaft was sunk at this point from which a little cupriferous material was taken. The copper appears to have been confined to the dike.

In the neighborhood of the large quarry, marked 4 on the geological-map, Fig. 3, there are large masses of magnetite scattered over the hillside, left by the weathering away of the softer country rock, the quartz-porphyry.

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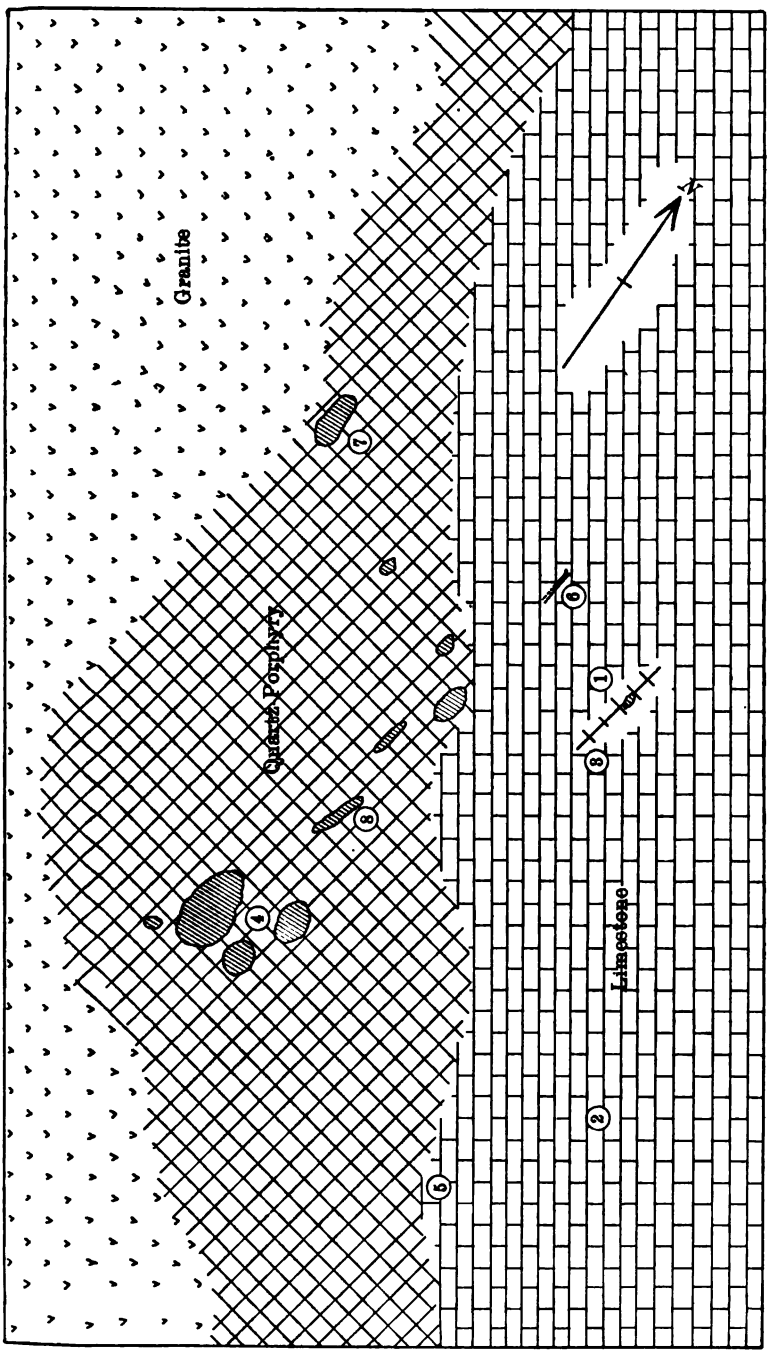

FIG. 3.—GEOLOGICAL MAP OF SHOOTS OF GARNET-ROCK AND PRINCIPAL FORMATIONS. THE SCALE IS 1 IN. EQUAL TO 200 FT., VERY NEARLY.

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2. *Occurrence of the Copper.*

The copper ore-bodies are found solely within the area of the quartz-porphyry; instances of their occurrence in the other rocks are wanting.

The porphyry does not carry copper throughout, the mineralization occurring in irregular and unconnected masses limited in area, but apparently continuous to unknown depths These mineralized channels or chimneys occur with greatest frequency within 300 ft. of the limestone contact, but the largest separate mineralized area is found rather nearer the granite than the limestone.

The outcrops of the mineralized portions of the eruptive as developed are shown on the surface-map, Fig. 2, where they are represented by the shaded areas.

3. *Granite-Contacts.*

The contact of the granite with the quartz-porphyry may be traced on the surface for a long distance, but the weathering of both rocks has so obscured the boundary that the only clear exposure is in the mine. Here the porphyry becomes felsitic along the junction, and carries for some distance back unaltered inclusions of the older rock. The contact is sharp and well defined, but as the two rocks are separated from each other by a seam of gouge the question of its being a fault-contact is at once raised.

A direct contact between the granite and the limestone is said to have been exposed in the workings of the old Grand Prize mine a short distance to the east of the cupriferous deposits. These workings are now inaccessible, and although the property was visited, but little of a geological nature could be ascertained. The ores are said to have been wholly of lead, copper being absent.

It is unfortunate that the contacts of the granite with the other rocks have not been better exposed.

IV. Descriptions of the Rocks.

1. *Granite.*

The extent of the granite has already been noted, and it is believed to be the fundamental rock of the district, all the evidence pointing to this conclusion. This cannot be stated with

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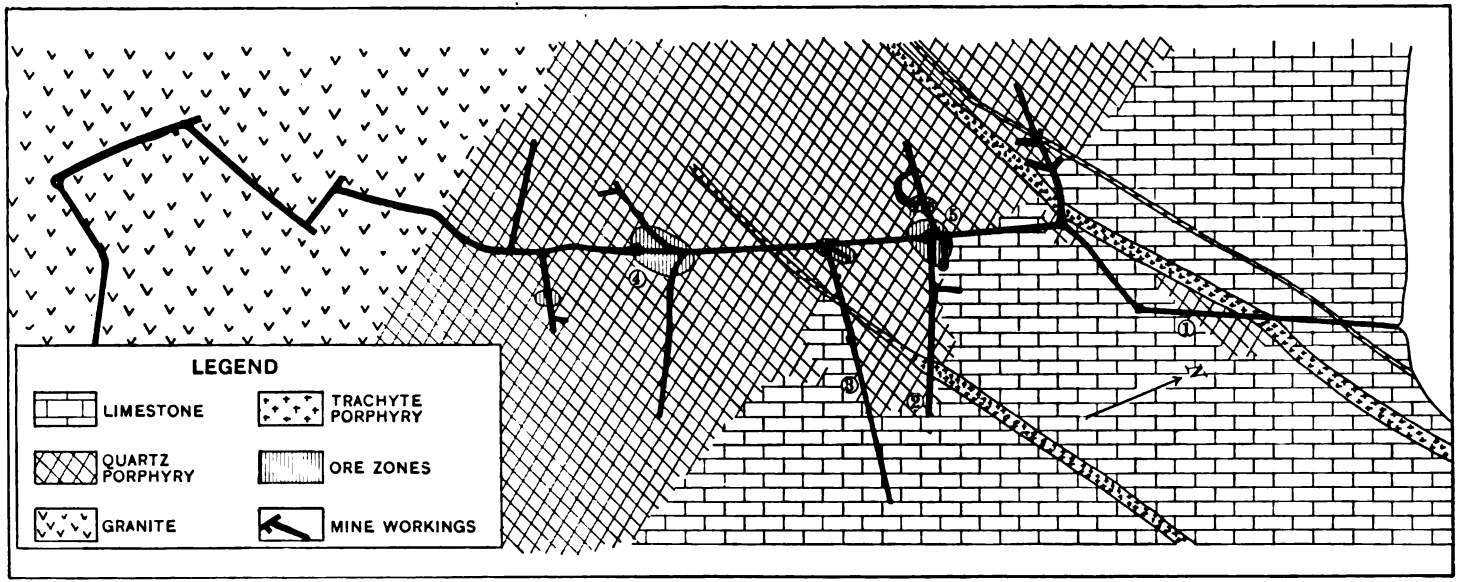

FIG. 4.—HORIZONTAL GEOLOGICAL SECTION ON THE LINE OF THE 700-FT. TUNNEL (ALBERT TUNNEL).

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certainty, however, as the geological investigations were not carried out over a sufficiently extended area to produce conclusive evidence.

The granite is a coarsely-crystalline rock revealing to the unassisted eye orthoclase and quartz with a little biotite; the dark silicates are present in very subordinate quantity or are lacking. Under the microscope the same minerals appear as those just mentioned. Some plagioclase is also apparent, and there is a rather decided tendency toward a coarse porphyritic texture. Larger feldspars and quartzes stand out in comparison with others smaller and with the dark silicates.

The granite exposures where studied were of very uniform texture and show no evidence of foliation or other effects of dynamic action, such as are frequently met in the older rocks of this class.

As previously stated, the only ores which occur in the granite carry lead. Copper has nowhere been found in it to our knowledge. The granite weathers easily and crumbles to a coarse sand. In the mine the workings in it are always wet, and the rock disintegrates and swells under the action of the water.

2. *Limestone.*

The limestone in typical specimens is a fine grained bluish rock, though in places in the mine it is almost black from included carbonaceous material. During the driving of several of the cross-cuts through this rock sufficient gas was given off to pollute the air in the workings.

The bedding-planes are not well defined; their prevailing dip is to the east at varying but usually high angles. Bands of chert several feet in thickness are of prominent occurrence, standing out in high relief on the hillsides.

Two analyses have been prepared by J. F. Kemp, from specimens taken near the mouth of the Albert Tunnel. No. 86 was 50 ft. in and was a blue unmetamorphosed variety. No. 96 was farther in and 45 ft. from the contact with the porphyry. It is a white crystalline marble.

	No. 86	No. 96
SiO_2,	14.77	1.38
Fe_2O_3, Al_2O_3,	3.68	1.01
$CaCO_3$,	72.53	96.01
$MgCO_3$,	8.89	1.36
Total,	99.87	99.76

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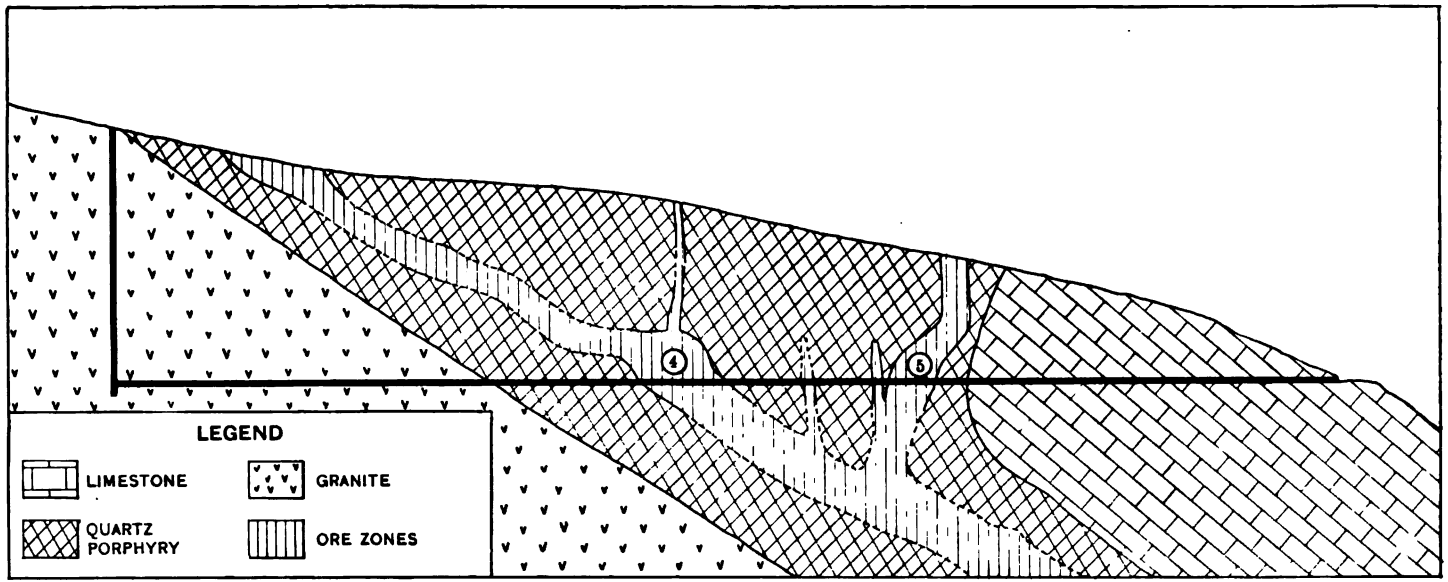

FIG. 5.—VERTICAL GEOLOGICAL SECTION, APPROXIMATELY ON THE LINE OF THE ALBERT TUNNEL.

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These analyses show considerable variation, but on account of the peculiar position of the garnetized pipes, or chimneys, within the eruptive, the composition is of less importance than in the usual case of contact metamorphism.

3. *Porphyritic Eruptives.*

The porphyritic eruptives certainly represent two distinct periods of intrusion which yielded two rocks of contrasted appearance to the eye, although composed of much the same minerals. The older rock contains the ore-bodies and varies from a felsite to a granite-porphyry in texture. Masses that are really true granite are also met, but they may be either inclusions torn off from the large granite mass, earlier described, or else textural variations of the porphyry itself. Their occurrence is too irregular and limited to lead us to believe them separate intrusions. Inclusions of limestone appear with even greater frequency.

The second rock is of marked and uniform porphyritic texture, with abundant, large, white phenocrysts, in a gray, felsitic ground-mass. The phenocrysts weather out at times and are in many cases Carlsbad twins of orthoclase. No phenocrysts of quartz have been observed. This second rock forms dikes which penetrate both the older porphyry and the limestone. Its entrance is later than the formation of the garnet, and it simply marks the last eruptive activity in the region.

We shall refer to the earlier rock as quartz-porphyry and to the later one as trachyte-porphyry.

(*a*) *The Quartz-Porphyry.*—We use this general name for the rock because it varies in texture all the way from a finely-crystalline felsite, through rhyolite-porphyry to granite-porphyry. The component minerals are chiefly feldspars and quartz. The quartz often appears as large phenocrysts (see Fig. 6), but again this form may fail. When seen under the microscope the large crystals are often corroded and embayed, sometimes excessively so. The small quartzes in the ground-mass show a marked tendency toward cross-sections that are nearly square and that are evidently cut through small bi-pyramidal crystals. In several specimens from the Albert tunnel, near the granite, excellent micropegmatitic intergrowths with feldspar are met. (See Fig. 8.)

The feldspars are prevailingly acidic plagioclases, but ortho-

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clase is also quite abundant. We find large phenocrysts up to an inch in cross-section and very abundantly set in the rather inconspicuous ground-mass, and also small crystals of rectangular or irregular habit forming the ground-mass itself. (See Fig. 7.) The feldspars are in no way remarkable or exceptional and, except for the prevalence of the acid plagioclases, are what we often observe in granite-porphyries.

An analysis of a variety with a moderate abundance of feldspar, set in a finely felsitic ground-mass, is as follows. It was very kindly made for us by Mr. T. T. Read, now of the Department of Metallurgy, Colorado College: SiO_2, 68.43; Al_2O_3, 16.08; FeO, Fe_2O_3, 1.59; MnO, 0.26; CaO, 2.93; MgO, 1.15; Na_2O, 5.36; K_2O, 4.19; Moisture, 0.11; H_2O, 0.61; total, 100.71 per cent.

The analysis in one small particular is incomplete for recasting into the percentages of the component minerals, in that the FeO is not separately determined from the Fe_2O_3, but the amount is so small that no appreciable error is involved if we consider it all Fe_2O_3, and calculate it as biotite, of the formula K_2O, $4MgO$, $2Fe_2O_3$, H_2O, $6SiO_2$. There will then remain a few residues of CaO, MgO, and MnO, to be relegated to hornblende or augite, both of which have been often noted in other specimens of the quartz-porphyry. In any event the margin of error is very small. When recast under these conditions the results are as follows: Quartz, 15.40; orthoclase, 22.24; albite, 45.06; anorthite, 8.90; biotite, 4.75; CaO, SiO_2, 2.32; MgO, SiO_2, 0.80; MnO, SiO_2, 0.30, H_2O, 0.40; total, 100.17 per cent.

The rock is thus one in which the light-colored minerals greatly predominate.

In addition to the type of rock which was the subject of the analyses, and which is illustrated by the photomicrograph, Fig. 6, we have others which are coarser and which have a larger percentage of the dark silicates. The increasing coarseness of grain is shown in Fig. 7. The dark silicates are richer in hornblende and augite, while biotite tends to diminish. The variations are changes in amount rather than in kinds of minerals.

The supposed inclusions of granite are more basic than the quartz-porphyry and are rather finely granitoid in texture.

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They present the same minerals with the additional occurrence in one slide of a crystal of allanite.

The quartz-porphyry weathers rather easily along the crushed zones, rendering the keeping open of the stopes a difficult problem. The granitic facies of this rock appear to be more resistant to weathering than the porphyritic varieties.

In some of the hand-specimens when carefully studied small dikes may be observed, an inch or less in width, cutting other phases of the porphyry and seeming to be other and later intrusions. They are all so small and limited in size and extent and are so exactly the mineralogical composition of the normal porphyry that it is difficult to believe them other than portions which have forced their way while still fluid into crevices in the otherwise already solidified mass. Prof. W. S. Crosby years ago described similar phenomena in the dikes along the Atlantic Coast as "extravasated dikes." At White Knob they are very minor occurrences.

(*b*) *The Trachyte-Porphyry.*—The trachyte-porphyry occurs in dikes whose widths vary from a very few inches up to 50 ft., cutting through the limestone and the quartz-porphyry.

The texture is coarsely porphyritic, Carlsbad twins are abundant and quartz is visible in but very small quantity, in a felsitic ground-mass which becomes dark on weathering. This rock presents a striking appearance in the mine, the large Carlsbad twins contrasting strongly with the darker ground-mass. The distribution of the phenocrysts, due to flowage, is everywhere in evidence, especially along the edges of the dikes, where the crystals of orthoclase have been drawn out into fine lines.

On exposure to the air the trachyte-porphyry usually softens and swells. No copper or pyritic material has been noted in any of the dikes.

The microscope chiefly seems to corroborate the observations made with the eye alone. A little quartz is visible in the slides and plagioclase appears in greater amount than would be suggested by the Carlsbad twins among the phenocrysts. The relations are similar to those shown in the recasting of the analysis of the quartz-porphyry. The dark silicate is chiefly altered to chlorite, but it appears to have been biotite. No analysis has been prepared of this rock.

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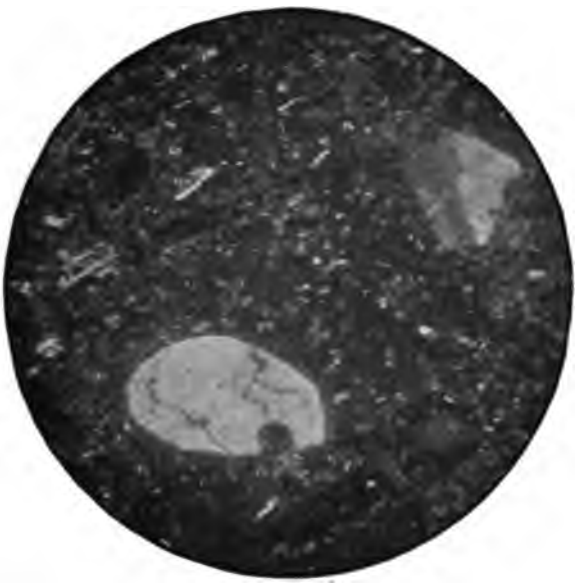

FIG. 6.—FINE-GRAINED QUARTZ-PORPHYRY, SHOWING AN EMBAYED QUARTZ BELOW; A PLAGIOCLASE CRYSTAL AT THE RIGHT ABOVE; AND SEVERAL PRISMS OF HORNBLENDE. CROSSED NICOLS; ACTUAL FIELD, 2.5 MM., OR 0.1 IN.

FIG. 7.—COARSE-GRAINED QUARTZ-PORPHYRY, WITH A ZONAL PLAGIOCLASE PHENOCRYST. CROSSED NICOLS; ACTUAL FIELD, 2.5 MM., OR 0.1 IN.

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FIG. 8.—MICROPEGMATITIC PHASE OF QUARTZ-PORPHYRY. CROSSED NICOLS; ACTUAL FIELD, 2.5 MM., OR 0.1 IN.

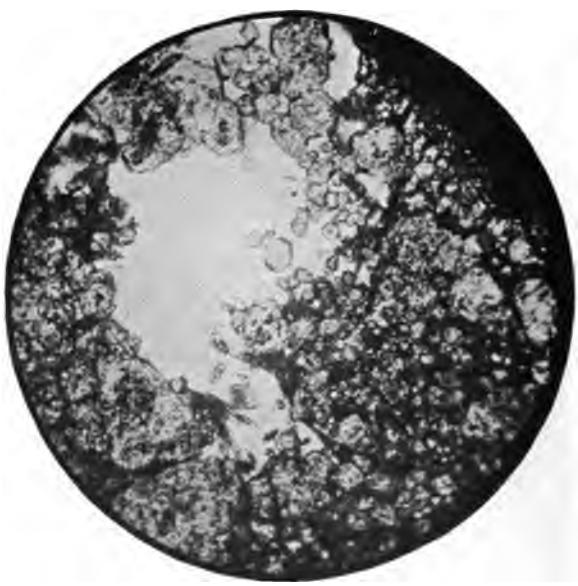

FIG. 9.—GARNETS. THE COLORLESS AREA IS QUARTZ. THE GARNETS BEGIN AS WELL-BOUNDED CRYSTALS, BUT BY MUTUAL INTERFERENCE BECOME IRREGULAR. ORDINARY LIGHT. ACTUAL FIELD, 2.5 MM., OR 0.1 IN.

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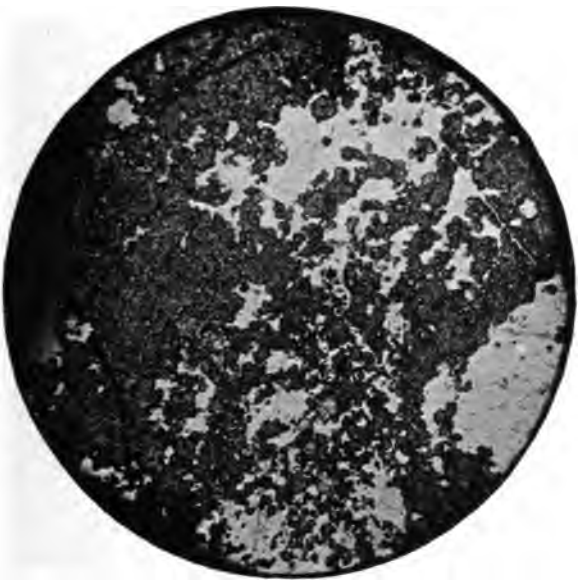

FIG. 10.—GARNETS AND DIOPSIDE TOGETHER IN COLORLESS QUARTZ. ORDINARY LIGHT; ACTUAL FIELD, 2.5 MM., OR 0.1 IN.

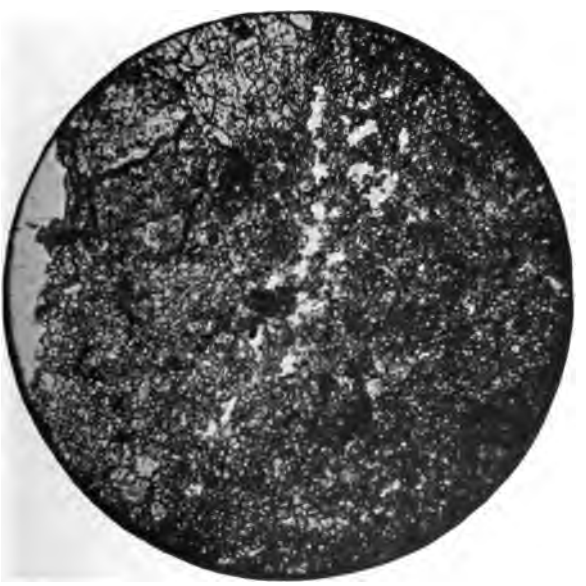

FIG. 11.—DIOPSIDE IN QUARTZ. ORDINARY LIGHT; ACTUAL FIELD, 2.5 MM., OR 0.1 IN.

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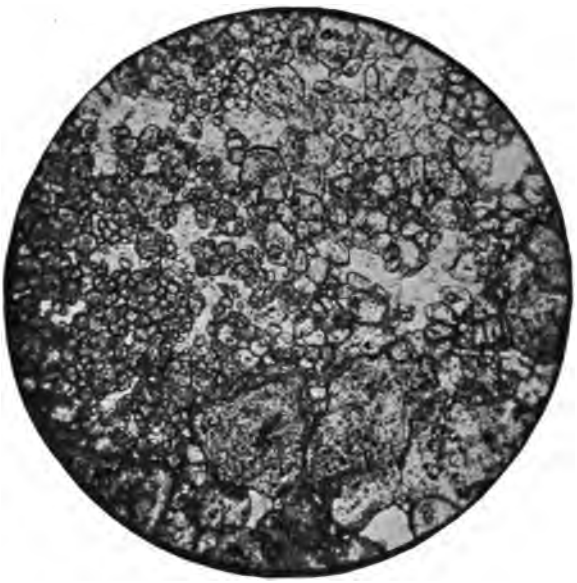

FIG. 12.—THE COARSER GRANULES ON THE RIGHT AND BELOW ARE GARNET; THE FINER ONES, DIOPSIDE. THE COLORLESS AREAS ARE CALCITE. ORDINARY LIGHT; ACTUAL FIELD, 2.5 MM., OR 0.1 IN.

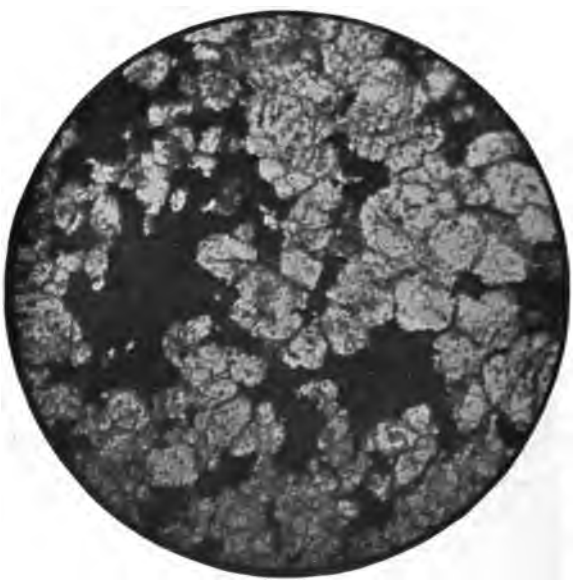

FIG. 13.—THE LIGHTER CRYSTALS ARE GARNET; THE BLACK AREAS ARE CHALCOPYRITE AND MAGNETITE. THE FIGURE ILLUSTRATES THE INTIMATE INTERGROWTH OF THE THREE MINERALS. ORDINARY LIGHT ACTUAL FIELD, 2.5 MM., OR 0.1 IN.

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V. Contact Phenomena.

1. *Comparative Immunity of the Limestone from the Effects of Contact-Metamorphism.*

The comparative immunity from the effects of contact-metamorphism enjoyed by the limestone is remarkable. Along its boundaries with the cupriferous eruptive a few feet at most have been changed to a crystalline white marble, but instances have been noted where typical unchanged limestone carrying bituminous matter occurs within a few inches of such contacts.

It may be stated generally that alteration of the limestone has proceeded to a greater degree along those contacts with the quartz-porphyry where the eruptive carries copper-values than where it is unmineralized.

The effect of the intrusive trachyte upon the limestone has been even slighter than that of the quartz-porphyry. Many instances have been noted where the limestone can be traced to a contact with this rock absolutely without visible change.

2. *The Contact Phenomena in the Quartz-Porphyry.*

As is the usual experience, the simplest contact-effect is the change of the blue limestone to white marble. The next effect is the production of tremolite. In one of our specimens this forms a vein about 2 in. thick of fibrous or acicular crystals in apparently unchanged blue limestone. The acicular crystals radiate like spherulites from a center and are individually as much as an inch in length. The tremolite was undoubtedly formed as a replacement-vein by the circulation of heated siliceous waters or vapors along a small original crevice in the limestone. Its production left the walls of blue limestone practically unchanged. Similar phenomena are reported by Lindgren from Morenci, Arizona.[1]

Along the contact of the quartz-porphyry and the limestone, garnet rock is not of prominent development. Its almost exclusive occurrence is within the eruptive mass itself, with which it makes a very complex intermixture. The mineralogy and petrography of these lime-silicate rocks will first be set forth, after which the subject of their development will be taken up.

[1] *Professional Paper* No. 43, *U. S. Geological Survey*, pp. 160, 161 (1905).

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The chief mineral is garnet. This varies from a light amber-colored or pale yellow translucent variety, whose composition is shown in analysis No. 1 of Table I., through various shades of pale green and brown to deep reddish brown and almost black. Crystals of all sizes, up to an inch in diameter, have been developed in cavities and are often coated with calcite or chalcopyrite. The forms are the usual combination of the rhombic dodecahedron and the tetragonal trisoctahedron, sometimes one, sometimes the other form predominating. The very beginnings of small garnets are shown in Figs. 8, 9, and 10, and the growth of larger ones is illustrated in Fig. 13. In each case they develop well-bounded crystals until in growth they interfere with one another.

TABLE I.—*Analyses of Garnet.*

	1	2	3	4	5
SiO_2,	37.07	37.79	37.15	42.63	36.26
Al_2O_3,	17.42	11.97	6.98	1.53	0.78
Fe_2O_3,	10.81	15.77	19.40	31.41	32.43
FeO,	0.68	1.31	.....	0.30	0.32
MnO,	.....	0.31	.....	0.43	0.27
CaO,	32.77	32.57	32.44	23.37	29.67
$CaCO_3$,	.....	.....	4.20	None.	None.
MgO,	0.51	0.37	.....	None.	None.
H_2O_-	0.14	0.09	.....	.....	0.13
H_2O+	0.39	.....	.....	.....	0.44
Soluble Fe_2O_3, Al_2O_3, etc.,	.....	.....	0.43	.....	.....
Total,	99.79	100.18	100.60	99.67	100.30

1. Light amber-colored garnet, White Knob, Idaho, by Cyril Knight.
2. Massive garnet, White Knob, Idaho, by T. T. Read.
3. Garnet, San José, Mexico, J. P. Kemp, *Trans.*, xxxvi., 192 (1906).
4 and 5. Garnet, Morenci, Ariz., W. Lindgren, *Professional Paper* No. 43, *U. S. Geological Survey*, 134.

Much more abundant than the crystalline or distinctly granular garnet rock, is a dense, often aphanitic variety with a peculiar greasy luster. Its garnet character might not be suspected at first sight, but under the microscope one sees that it consists of this mineral with almost nothing else. An analysis of a typical specimen is given under No. 2. Three garnets from the contact-zones of other localities are added under Nos. 3, 4, and 5.

All of the analyses in Table II. have been recast in order to

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discover and emphasize the variety of garnet which occurs in the zones. The prevailing impression is that grossularite, the lime-alumina variety, is the principal and characteristic one. Increasing experience shows, however, that this is a mistake and that andradite, the lime-iron molecule, is much more widespread than has been generally believed. So far as the determinations have gone, we cannot but be impressed with the small percentages of FeO, MnO, and MgO, all of which oxides enter in only a subordinate way.

TABLE II.—*Analyses of Garnet Recast from Table I.*

	1	2	3	4	5
Grossularite.					
$3CaO, Al_2O_3, SiO_2$, .	69.26	47.82	31.00	6.02	2.96
Andradite.					
$3CaO, Fe_2O_3, 3SiO_2$, .	21.13	44.16	61.10	67.62	86.30
Almandite.					
$3FeO, Al_2O_3, 3SiO_2$, .	1.61	2.99			
Pyrope.					
$3MgO, Al_2O_3, 3SiO_2$, .	1.44	1.31			
Spessartite.					
$3MgO, Al_2O_3, 3SiO_2$, .	.....	0.68	.....	0.91	0.79
Hematite, . . .	4.25	1.90	.....	7.30	4.80
Magnetite, . . .	.....	.....	.....	1.00	1.00
Kaolin, ...	2.61	.....	1.04		
Wollastonite.					
CaO, SiO_2.					
Quartz, ...	.....	1.32	2.58	17.46	4.26
Calcite, ...	.....	.....	4.20		
Water, ...	.....	0.09			
Total, . . .	100.30	100.27	99.92	100.31	100.11
Excess of SiO_2 . . .	0.54				
	99.76				

In recasting the last two analyses, in Table II., which are taken from Mr. Lindgren's paper,[2] the calculated mineralogy varies in some particulars from the one observed and recorded by the author. No magnetite, for example, is mentioned by him, and the quartz is perhaps in excess of what would be inferred from his descriptions. But the great point is not affected—viz., to show the kinds and relative amounts of the several garnet molecules.

Next after garnet, diopside is the most abundant mineral. It appears in finely granular aggregates of brightly polarizing

[2] *Professional Paper No.* 43, *U. S. Geological Survey*, p. 134 (1905).

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properties. In the hand-specimen the grayish finely-crystalline rock often corresponds perfectly to the old name, lime-silicate-hornstone, and might be taken for a felsite, but a little experience with the microscope corrects the impression that it is an eruptive. The very beginnings of diopside in finely-granular masses are shown in Fig. 11. Its simultaneous growth with garnet is illustrated in Figs. 10 and 12. The following analysis by T. T. Read, to whom grateful acknowledgments are due, illustrates the chemical composition. It has been recast with the results which follow:

Diopside rock.	Per Cent.	Quantity of Minerals.	Per Cent.	
SiO_2,	45.85	Pyroxene:		
Al_2O_3,	12.21	CaO, SiO_2,	59.15	Diopside.
Fe_2O_3,	2.15	MgO, SiO_2,	8.60	Diopside.
FeO,	2.49	FeO, SiO_2,	4.59	Diopside.
CaO,	28.54	MgO, Fe_2O_3, SiO_2,	2.91	Augite.
MgO,	8.70	MgO, Al_2O_3, SiO_2,	24.21	Augite.
Total,	99.94	Quartz,	0.12	
		Total,	99.58	

The analyses indicate 72.34 per cent. of diopside, strictly speaking, and 27.12 per cent. of the augite molecule. The pyroxene is therefore not pure diopside, but only predominantly composed of this mineral.

Wollastonite has been detected in several specimens, but it is not an abundant mineral in the zones. Its molecule, CaO, SiO_2, forms nearly 60 per cent. of the specimen of diopside whose analysis is given above, but there is enough of the other molecules to destroy the identity of the wollastonite itself.

Vesuvianite and epidote, both closely related to garnet in composition and characteristic components of garnet zones, have been identified but are rare. The former is $5CaO, MgO, Al_2O_3, AlOH, 5SiO_2$, and the latter $2CaO, Al_2O_3, AlOH, 3SiO_3$. The formulas will make clear that their occurrence in the zones is entirely natural and to be expected, and it is perhaps surprising that they are so rare.

Scapolite, a mineral not especially abundant in the zones associated with copper-deposits in the West, has been once detected in the slides. It is a lime-alumina silicate which involves also some sodium and chlorine. It is so complicated that on

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account of its rarity it will not be discussed in greater detail. Its composition is also one to make it a very natural member of the contact-zone group.

Fluorite, the calcium fluoride, appears in quite large amounts in the upper and branching portions of the deposits. It is mingled with magnetite, chalcopyrite, the light, amber-colored garnets, and calcite, and is a very natural product of the pneumatolytic processes which produce the contact effects.

Calcite is widely distributed, both among the other components of the zones and as crusts and veinlets more recently formed than they.

Gypsum has also been observed as a mineral of late development, probably produced by the alteration of the sulphides.

Quartz is of wide distribution, although not in large amount. It either forms veinlets through the others or crusts of clear crystals.

Metallic minerals are not numerous or greatly varied. The original ones include magnetite and specular hematite, pyrite, chalcopyrite, bornite, and rarely a little zinc-blende and galena. Among the secondary minerals chrysocolla is much the commonest of those containing copper, although malachite and chalcocite are not unknown. Limonite results from the oxidation of the pyrite and chalcopyrite, and may so richly contaminate the chrysocolla as to yield the brown resinous variety of the latter, the *Kupferpecherz* of the German miners.

The order of formation of the several minerals seems to be the following: Garnet, diopside and probably the related silicates, vesuvianite and epidote, were the first, and where they could grow freely they developed well-bounded crystals. Before their period closed, magnetite, chalcopyrite, pyrite and specularite began, since the latter are found as inclusions in the silicates. (See Fig. 13.) But the sulphides certainly continued longer than the garnet, because they incrust its well-developed crystals. Bornite appears to be a later mineral, since in one or two specimens it forms veins cutting the garnet rock. It has been found mottled with inclusions of chalcopyrite, but whether it has resulted from the enrichment and replacement of earlier existing chalcopyrite or is itself original is difficult to decide.

In the alteration and secondary enrichment by far the com-

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monest mineral to form has been chrysocolla, and it does not appear to have moved far from its parental chalcopyrite. As will be more fully brought out later, it seems as if the abundant silica in the minerals of the gangue locked up the copper in the hydrated silicate before appreciable migration had been accomplished. The relatively small amounts of calcite have, however, sufficed also to produce a little malachite. Chalcocite has been noted in but one place in the mine, and then in but small amount.

3. *The Form and Distribution of the Ore-Bodies.*

Both in form and in relations to the country-rock these ore-bodies are different from any which are known to us, and therefore much care has been exercised in preparing Figs. 2, 3, 4 and 5, which are intended to illustrate them and which are based on careful geological observations. The endeavor has been made to indicate what is known and what is inferred. The known boundaries are drawn in full lines, the inferred in dotted.

In Fig. 2 the surface geology is shown, and the profile is in some degree indicated by the section, Fig. 5. It is at once apparent that the ore-shoots outcrop altogether in the quartz-porphyry, except for one small one, which is associated with an outlying dike in the limestone at 1, Fig. 3. Ten shoots are shown in the quartz-porphyry, some near the contact with the limestone, others near the granite. In depth the tendency seems to be rather to approach the limestone than otherwise, and this is brought out in Fig. 5.

Each of the ten exposures on Fig. 3 marks a chimney of garnet rock whose course has been shown by the workings to be sinuous when followed downward. The chimneys also tend to come together with descent, but never, so far as known, to do the same with ascent. The northern group marked "5" on the map, Fig. 4, and illustrated more fully on Fig. 5, has been continuously opened by winzes, up-raises and stopes for 500 ft. from the surface to the level of the Albert tunnel, and we note that, as shown on this figure and on Fig. 4, the disposition of the chimneys to coalesce is pronounced. Again, if we take No. 4 of Fig. 4 and Fig. 14, of which the latter represents a

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section of the southern shoot, 60 ft. above the former, the coming together within this distance is noticeable.

In the inferential portion of the section, Fig. 5, the hypothetical coalescence of all the shoots into one parent trunk in depth is indicated and is believed to be highly probable.

At the surface the northern group of garnet-shoots appeared as a capping of limonite which carried a small percentage of copper as the red oxide. Druses of bright unaltered specularite occurred in cracks throughout the mass. Masses of limonite occasionally inclose nuclei of unaltered cupriferous pyrite. As we descend, chrysocolla becomes the copper-bearing mineral, and pyritic matter is encountered but a short distance below the surface. From this zone down no changes in the mineralogical combinations of the copper take place. The bulk of the ore is cupriferous pyrite in a gangue of garnetized and altered quartz-porphyry. Through this mass percolating surface-waters have developed channels of oxidized ores, consisting of chrysocolla and, in lesser quantity, malachite as stains through the brecciated eruptive. At or just below the 700-ft. or Albert tunnel-level, a split has taken place in this ore-body, a sheet of pyritic matter branching off from the main chimney and continuing upward away from it with a southerly trend. This ore-body has been opened below the tunnel-level through a winze; the ores are the same as those stoped from the upper levels, and of similar grade, both oxides and sulphides being present in different parts of the chimney.

In the winze on the tunnel-level above mentioned and at a point about 210 ft. above this level, limestone forms the northern limit of this ore-zone. It has been altered to a white marble where exposed in the stopes, but that the alteration has taken place for a few feet only has been shown wherever the limestone has been penetrated by a cross-cut.

The southern ore-chimney (No. 4 on Figs. 4 and 5) is rather larger at the tunnel-level than the northern chimney just described, but immediately above, it splits into two bodies of nearly equal size, which steadily diverge as they go up. In the eastern branch are found the fumaroles described two paragraphs below. The western branch splits into several smaller branches a short distance above its junction with the eastern chimney; the conditions obtaining 60 ft. above the tunnel-level

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are shown on the map, Fig. 14; within a vertical height of 60 ft. the one large chimney at the tunnel-level has split up into five parts.

The ores and their distribution in the northern and southern chimneys are essentially similar, with the exception that those of the southern chimney, although at a greater depth below the surface where opened, are the more completely oxidized.

The smaller ore-shoots shown on Fig. 3, and the surface exposures are, in so far as developed, entirely similar to the typi-

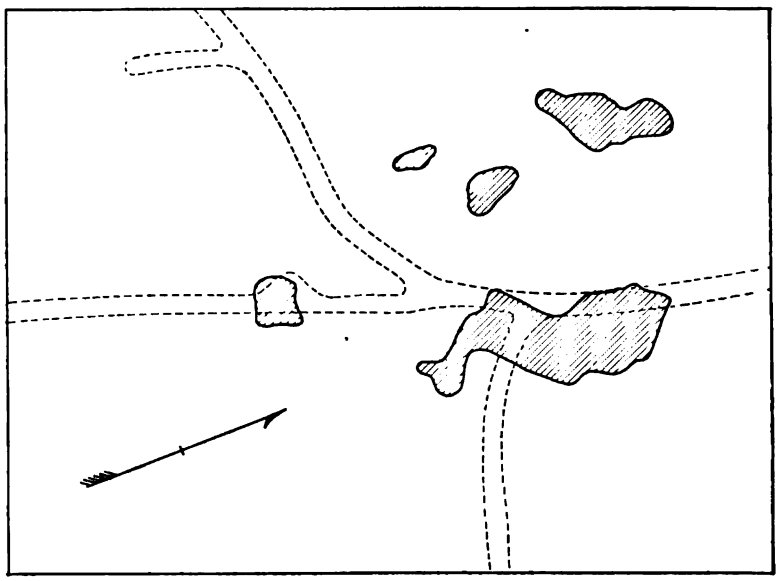

FIG. 14.—HORIZONTAL SECTION THROUGH THE SOUTHERN ORE-BODY, 60 FT. ABOVE THE ALBERT TUNNEL. THE DIVERGENCE OF THE SHOOTS WITH ASCENT IS SHOWN. COMPARE NO. 4 OF FIG. 4. THE DOTTED LINES ARE THE TUNNEL AND DRIFTS.

cal chimneys described, and a detailed account of them would only be a repetition of the foregoing.

On the 9th floor of the east stope on the ore-chimney marked 4 on Fig. 3, there has been exposed a cluster of what may very well be termed "pipes," or "flues." These are surrounded for perhaps a radius of 20 ft. at the point exposed by an intimate mixture of finely-crystallized garnet and specularite, which carries pyrite and chalcopyrite, and their products of oxidation, and fluorite in small but persistent quantity. Through

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this mass are vugs lined with crystalline garnet, and in several places angular fragments of unaltered quartz-porphyry cemented in the mass, bearing testimony to the fumarolic processes.

The pipes themselves are lined with incrustations of garnet and specularite. Crystalline purple fluorite is present and chalcopyrite occupies the spaces between the other minerals. Additional minerals of probably later origin are siderite incrusting the other minerals, gypsum, which occurs in a similar manner, and calcite, which has come in latest of all, filling the open spaces between the larger crystals and the cracks through the formation.

These pipes vary in size from those which are now closed, but which exhibit the original structure, to open channels from 8 to 10 in. across.

In the north drift on the second level from the shaft at the point marked 8 on the geological-map, Fig. 3, there is a similar occurrence. The resulting mineralization has not been over as extended an area as in the case already cited, but the two occurrences are essentially similar.

VI. The Method of Deposition.

In tracing the method of formation of these ore-shoots we have therefore to account for their cylindrical character, their sinuous and forking courses, and their limitation to the quartz-porphyry. The most reasonable and natural explanation is that they have been produced by the passage of highly-heated solutions or vapors, or ionized water-gas, through the quartz-porphyry, and probably while the latter was still viscous or not entirely solidified. The vapors, for such they would appear to have necessarily been, if the quartz-porphyry were still unsolidified, must have been highly charged with lime and iron. The lime and iron combined with the components of the porphyry to yield the garnet and diopside rock of composition shown by the analyses. We have sufficient data to roughly calculate if this change was probable, or at least to determine what new additions to the quartz-porphyry would be necessary. We must assume that the silica, which is already in excess in the quartz-porphyry, is not increased, but rather is brought down to the grade of the contact rocks by additions of bases. We

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also assume that the alkalies of the quartz-porphyry are eliminated, since they are not found in the garnet and diopside rocks. The fundamental analyses upon which the calculations are based are given in Table III., but several minor ingredients are omitted in recasting.

TABLE III.—*Change of Quartz-Porphyry to Garnet.*

	1	2	3	4	5	6	7
	Quartz-Porphyry.	Crystallized Garnet.	Massive Garnet.	Diopside.	Quantities added to 1 to yield 2.	Quantities added to 1 to yield 3.	Quantities added to 1 to yield 4.
	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.
SiO_2, .	68.43	37.07	37.79	45.85			
Al_2O_3, .	16.08	17.42	11.97	12.21	16.08	5.59	2.14
Fe_2O_3, .	1.59	10.81	15.77	2.15	19.75	29.59	5.75
FeO, .	.....	0.68	1.31	2.49			
MnO, .	0.26	.....	0.31				
CaO, .	2.93	32.77	32.57	28.54	57.57	56.04	39.66
MgO, .	1.15	0.51	0.37	8.70	.....	.....	11.83
Na_2O, .	5.36						
K_2O, .	4.19						
Moisture,	0.11						
H_2O, .	0.61						

In these determinations the MnO, and MgO of Nos. 2 and 3 were rejected, as they amount to but little. Since FeO was not determined in No. 1 all the FeO respectively of Nos. 2, 3 and 4 was recast as Fe_2O_3, and was added to the Fe_2O_3 of the analyses, and these totals were then used. The small components make very little difference at best. The calculations are based upon continuous proportions to which the silica of the eruptive gives the clue.

Thus for the crystallized garnet, No. 2:

68.43 SiO_2 : (16.08 + 16.08) Al_2O_3 : (1.59 + 19.75) Fe_2O_3 : (2.93 + 57.57) CaO.

as 37.07 SiO_2 : 17.42 Al_2O_3 : 11.56 Fe_2O_3 : 32.77 CaO,

or, as 37.53 SiO_2 : 17.64 Al_2O_3 : 11.70 Fe_2O_3 : 33.18 CaO.

Should a reader verify these, it must be borne in mind that the summation of the values in the middle member of the continued proportion does not total an even hundred, and allowance must be made for the missing percentages. Thus, as the summation is 98.82, we must raise each by practically 1.2 per cent., giving the third member, which figures were used. It should also be added that in recasting analyses 3 and 4, so as to obtain Nos.

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6 and 7, other but analogous proportions are required. After raising in Nos. 3 and 4, the FeO to Fe_2O_3, the second member of the proportions adds up so nearly 100, that no correction was esteemed necessary.

Solutions which would furnish the required accretions for Nos. 5, 6 and 7 would have their dissolved solids in the following proportions:

	5a	6a	7a
Al_2O_3	17.2	6.13	3.61
Fe_2O_3	21.2	32.44	9.68
CaO	61.6	61.43	66.79
MgO	.....	.....	19.92
	100.00	100.00	100.00

The alumina is the only component which presents any particular difficulties, for the others are very common ingredients of mineral springs. Thus, Berzelius[3] found the following results for the *sprudelstein* of the Carlsbad springs, whose waters are now considered by some observers to be magmatic in origin. In a general way the composition is closely akin to the ones required above.

	Per Cent.
$FeCO_3$,	12.13
Fe_2O_3,	19.35
$CaCO_3$,	53.20
Basic iron phosphate,	1.77
Alumina phosphate,	0.60
SiO_2,	3.95
H_2O,	9.00
Total,	100.00

Beyond this coincidence in compositions it does not seem possible to go in the way of positive evidence. We must therefore build up our conceptions by imagining the emissions of suitable compositions passing upward through the substance of the quartz-porphyry and yielding the chimneys.

The question may be raised, whether the mineralization may not have been produced along fissures in the already solidified porphyry and by heated waters. Fissures have been sought with great care in the mine and many joints have been plotted. But no coincidence can be traced between them and the garnet chimneys; much less can any intersection of fissures be shown

[3] Quoted in Roth's *Allgemeine und Chemische Geologie*, vol. i., p. 579 (1879).

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where the chimneys exist. The conclusion has therefore been reached that the uprising gases and vapors passing through a molten or viscous mass have at least established the lines for the development of the garnet, diopside and other minerals. If the necessary ingredients were contributed to a still-molten rock, the chimneys must have later solidified as masses of garnet and diopside when the temperature fell.

The passing of solutions undoubtedly continued after the consolidation of the garnet-zones, since we find fluorite and even quartz filling the interstices in the other minerals.

When the chimneys had been formed, elevated to their present position, and subjected to the meteoric waters within the vadose region, the alterations took place which have led to the production of the chrysocolla as the chief result. Once the copper passed into solution in the oxidation of the sulphides, it seems to have combined with silica to give the hydrated silicate and to have remained near its source. Waters now trickle downward in the stopes and winzes at numerous places. They have been drunk freely by the miners with no ill-effects. Samples were carefully gathered in many parts of the mine and tested for copper by Mr. T. T. Read with the most delicate reagents. No discernible amount of copper could be detected. There seems no reason, therefore, to infer the existance of a mass of enriched ores in depth.

To the student of the specimens as well as to one who goes over the map without experience on the ground, the hypothesis may suggest itself that the granite is the intrusive and that the quartz-porphyry is its border facies, chilled into the denser texture by contact with the limestone. Observation on the spot, however, has led to the conclusion that the granite is a separate and distinct rock from the quartz-porphyry, and that it is older.

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The Extraordinary Faulting at the Berlin Mine, Nevada.

BY ELLSWORTH DAGGETT, SALT LAKE CITY, UTAH.

(New York Meeting, April, 1907.)

THE Berlin gold-quartz mine is situated in Nye county, Nevada, on the west flank of the Shoshone range, about 40 miles south and 30 miles west from the town of Austin, the county-seat of Lander county. The distance from Austin is about 60 miles by stage-road.

The outcrop of the vein, at the top of the incline-shaft, is situated just at the base of the mountain proper, almost exactly at the intersection of the mountain-side with the gravelly bench that slopes for about three-quarters of a mile to the flatter sage-brush plain, or desert valley, below.

The vein itself consists almost entirely of quartz, with perhaps 2 per cent. of sulphide of iron, copper, lead, zinc and antimony, and perhaps a trace of some of the compounds of tellurium with gold and silver, although none of the latter have been as yet positively identified.

The relative proportion, by weight, of silver and gold in the ores, varies in different parts of the vein, from 12 silver for 1 of gold, to 7 silver for 1 of gold.

The quartz vein-filling is usually frozen fast to the walls, and is very hard and compact, seldom showing the friable, fissured, or shelly structure often to be found in quartz-veins.

Comparatively little evidence of relative motion of one wall of the vein upon the other is to be found—a fact indicating that during the formation of the vein, and prior to the extensive movements herein described, little disturbance had taken place. No evidence whatever of metasomatic origin has been observed. On the other hand, occasional occurrences of comby structure, in which the axes of the quartz-crystals are at right angles to the plane of the vein, rather indicate deposition from solution in a pre-existing fissure.

Spurs, or branches, and small parallel veins, while not en-

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tirely absent, are not thus far very numerous, and not extensive enough to possess any marked practical importance.

The thickness of the Berlin vein varies from a few inches to 8 ft., but, over far the greater portion of the explored area, is tolerably uniform at from 2 to 3 ft., measured normal to its plane. The average thickness of quartz thus far stoped, as determined by all available measurements, is a little less than 2.5 feet.

The course of the vein is NE. and SW., and its average dip about 45° to the SE.

The Berlin vein, prior to the extensive faulting described in this paper, was tolerably uniform in size, course and dip, and perhaps, on the whole, rather more regular than the average of gold-quartz veins.

It appears to have been a fissure filled with quartz, and may be said to have originally been in shape, structure and origin, a typical "old-fashioned" fissure-vein of the books.

Throughout the entire field covered by the underground works of the Berlin mine, the rock is andesite, which is, however, in places, locally so altered by compression or movement as to change considerably its appearance and structure. Some very limited chemical changes may also have occurred in places, by reason of which the above-given classification might, to a small degree, fall short in completeness.

The underground workings, including the stopes, of the Berlin mine, as existing July 1, 1906, are shown in plan in Fig. 1, which is a reduced copy of the working-plan of the mine, from which, for the sake of plainness, most of the survey-lines, station-numbers and heights above the datum-plane have been omitted.

The stopes, shown by the shaded areas, have in general the form of a more or less irregular parallelogram, suggesting at a glance the extensive faulting in two directions, to which the vein has apparently been subjected.

The ore-bodies, properly enough called segments, are usually terminated on all sides by fault-planes. Those on the east and west sides, though just as truly fault-planes, have been locally called breaks, which term will be retained in this paper for the sake of identification.

The lines bounding the segments, as projected on the plane

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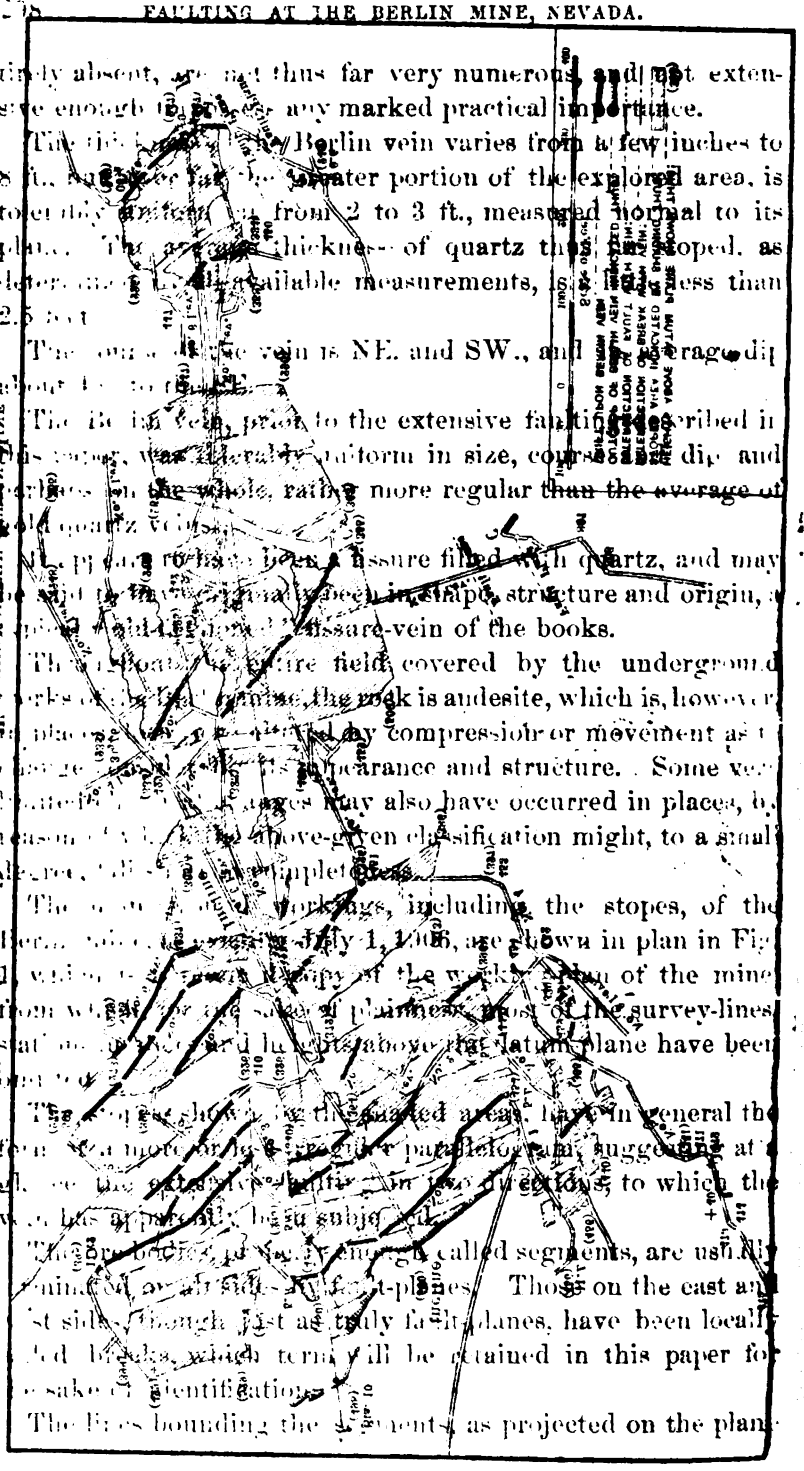

FIG. 1.—PLAN OF UNDERGROUND WORKINGS OF THE BERLIN MINE.

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The thickness of the Berlin vein varies from a few inches to 8 ft., but over far the greater portion of the explored area, is tolerably uniform at from 2 to 3 ft., measured normal to its plane. The average thickness of quartz thus fine-stoped, as determined by all available measurements, is a little less than 2.5 feet.

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The Berlin vein, prior to the extensive faulting described in this paper, was tolerably uniform in size, course, [illegible] dip and [illegible] on the whole, rather more regular than the average of gold quartz veins.

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The lines bounding the segments, as projected on the plane

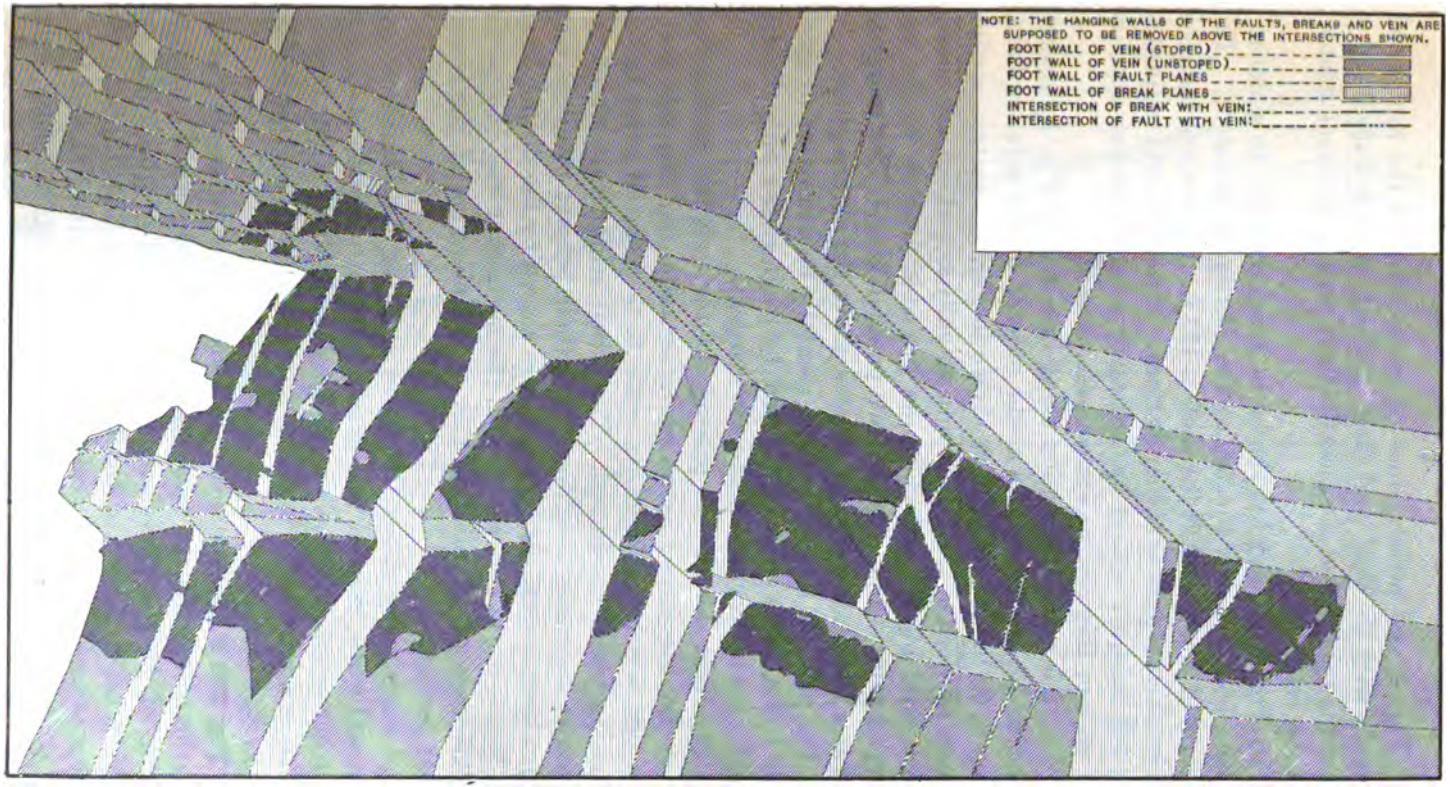

FIG. 1A.—STRATIGRAPHY OF THE BERLIN MINE.

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of Fig. 1, are mainly lines of intersection of the faults and breaks with the vein, and their projections do not at all represent the true course of either the fault-planes, the break-planes, or the vein.

In Fig. 1, the heights shown in brackets refer to a datum-plane 500 ft. below the top of the Berlin incline-shaft. The height of No. 8 level at the shaft is 137 ft. Figures not inclosed in brackets represent survey-stations. The stopes, where limited by full lines, are not cut by faulting fissures, but end on account of poor or thin ore, or some similar reason.

Fig. 2 is a vertical section along line *A B* of Fig. 1 through Station 10, on the surface at the outcrop of the vein, and Station 0, on the No. 6 level, as shown in Fig. 1. In this section those portions of the Berlin vein actually stoped out are shown as a solid black line, while the probable position of the unstoped vein is indicated by two parallel lines. The marginal figures in Fig. 2 show the heights above the datum-plane.

The plane of this section was carefully chosen so as to avoid the faults, and Fig. 2, considered by itself, shows only the disturbances apparently due to the breaks.

The light dotted line passing out through the surface-line may be considered as an elevation, showing the minimum heights which the segments could have occupied prior to the faulting herein considered. The actual height from which the present segments have dropped to their present position may have been several times as great, as shown by the dotted lines.

Nor is it yet certain whether the movement was due to the subsidence of the northwest or to the elevation of the southeast portion.

Among many sections made in studying the Berlin underground work, there is one nearly parallel to section *A–B*, but further north, which shows it possible to drive a flat incline-shaft, straight in line and grade, that would cut the vein no less than eight times.

Fig. 3 is a horizontal section, showing the intersection of the Berlin vein and of the break- and fault-fissures with the average plane of the No. 4 level, the average height of which is about 165 ft. below the top of the incline-shaft, or 335 ft. above the datum-plane. The true course of the vein—viz., NE. and SW.; of the breaks, nearly always N. and S.; and of the faults, about

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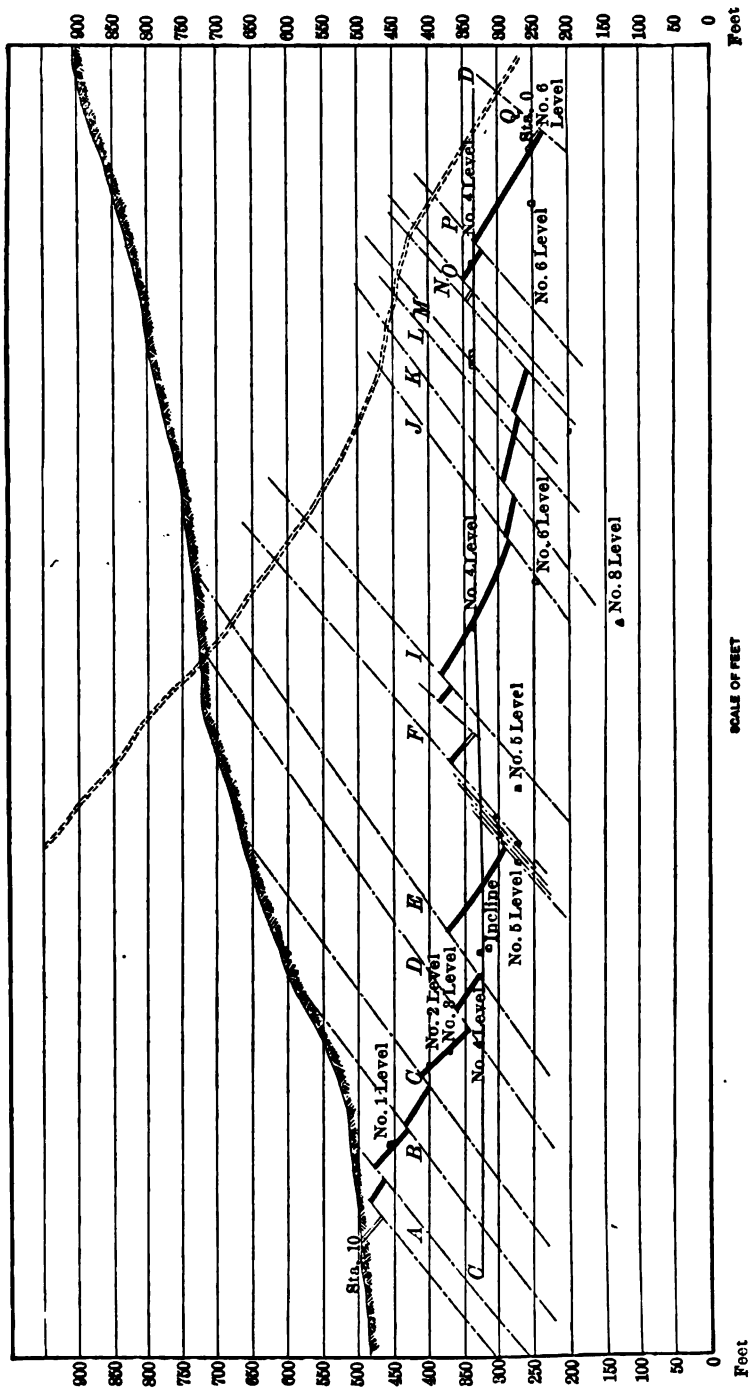

FIG. 2.—VERTICAL SECTION ALONG LINE *A B* OF FIG. 1.

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N. 60° W., is therefore correctly shown in Fig. 3, which, moreover, represents that portion of the fissure system which may be regarded as best known from the present developments.

If we imagine that, in the field covered by Fig. 3, the accidents of erosion had left the surface about at the plane of the No. 4 level, then the heavy black lines would represent the actual surveyed outcrop of a single vein that, before the faulting here described, was probably as regular, as uniform in strike and dip, as nearly in a true plane, and generally as free from eccentricities as the average quartz-vein.

So far as my observation goes, this situation is without parallel in quartz-mining.

It was at first supposed that the N. and S. fissures or breaks, dipping about 45° W., had first been formed and had faulted the pre-existing vein, and that subsequently a pair of fissures had occurred, each with one or more branches, having a general course about N. 67° W. and a dip of 63° N. 22° E., cutting and faulting both the vein and the breaks. But as developments progressed, and additional intersections of the breaks and faults were found, or indicated, it was observed that in several instances the faults were cut and faulted by the breaks.

In the case of the so-called north fault, shown in Fig. 1, the segments of the Berlin vein, from its outcrop for a distance easterly of about 1,300 ft., have been cut off. It was for a time believed that the south branch of this fissure was a continuous fissure, cut by the No. 4 level at its east end, near survey-station 450, and in the north branch of the No. 4 level, near station 459, and again near station 454. But a consideration of the position of the surveyed and known lines of intersection of the fault with the vein forming the north boundaries of the stopes, shows that these intersecting lines were very far from being in the same plane, and that a single fissure, to have contained them all, would have had to be extremely—in fact, impossibly—crooked and irregular.

The conclusion was therefore forced that some of the larger breaks had faulted also the north fault, as well as the vein, and that, instead of one continuous fissure, with a course N. 67° W., there were several fissures with an average course of about N. 60° W., and a dip N. 30° E., of about 63° from the horizontal.

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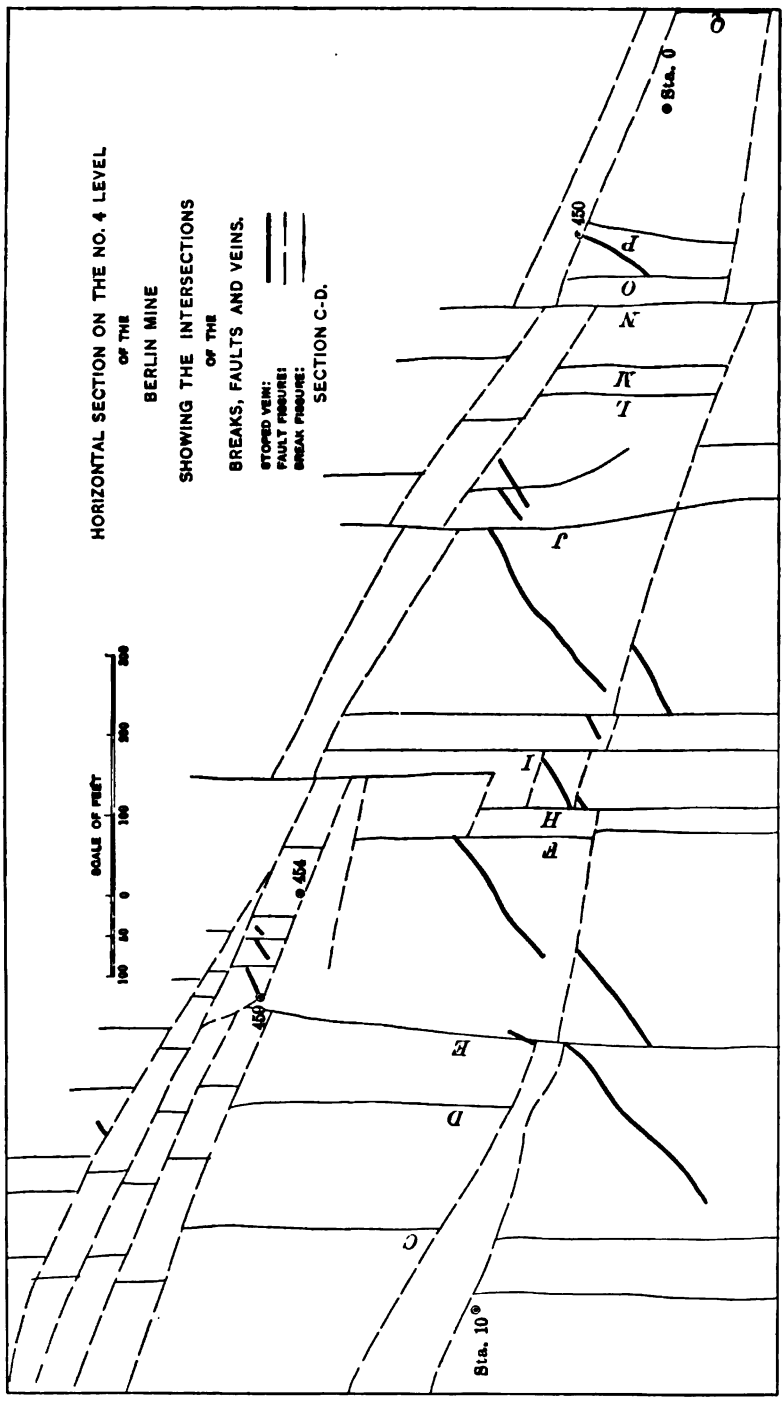

FIG. 3.—HORIZONTAL SECTION ON NO. 4 LEVEL.

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In practical mining, the main object is, of course, to find and extract the ore as cheaply as possible. It is not often that exposures interesting stratigraphically are incidentally made, or that special work for such a purpose is warranted. In the present case, while in many, perhaps in a majority of instances, the faults do cut and fault the breaks, there is no such uniformity as would enable us to establish the relative age of the two fissure systems. In fact, there are enough instances actually exposed, or undoubtedly indicated, of the breaks cutting and faulting the faults, to make it tolerably certain that the fissures of both systems originated at the same time, and in all probability from a single force.

If we assume that the planes of the break-fissures were in fact parallel to each other, and that the same was true of the fault-planes; that every fissure of each system has been cut and faulted by at least one fissure of the other system, and that the material fissured was rigid, incompressible and inelastic, it would appear that the line of any movement produced by gravity, or by an uplift from any cause, would necessarily be in both planes, and therefore in the line of the intersection of the two planes.

Now, in fact, the planes of neither the breaks nor the faults are exactly parallel. It is not known that every plane of each system has been cut or faulted by one or more planes of the other system. Moreover, andesite is far from rigid, being compressible, and capable of great distortion. Just so far, however, as the conditions existing in the Berlin field approach the hypothetical conditions outlined above, might we expect the direction of the movement to conform more or less closely to the direction of the intersection of the average fault- and break-planes.

Fig. 4 represents a very interesting occurrence, having a significant bearing on this point, which was recently uncovered by the accidental scaling-off of some slabs of clay and gouge in the northwest corner of the second large stope from the east end of the mine. The ore was stoped up to break *J*, but, in the north corner, not quite to the fault-plane—perhaps to within 4 or 5 ft. of it. At point *C*, 12 or 13 ft. along the break southwesterly from the little corner of remaining quartz, there was plainly exposed in the hanging-wall of the break the edge of a strong, faulting fissure, striking about N. 60°

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W., and dipping 60° northerly. In this fissure, with some coarser material, was a layer of about 1 ft. of stiff blue clay, evidently the product of attrition. This layer of clay, without any parting whatever, and about uniform in thickness, was continuous around the sharp angle into the break, and up in the break to the quartz remnant, precisely as a layer of lubricant might be found in the V-groove of a planer. It no doubt continues up to the fault-plane on the north, and there turns down into it. Just southeast of point *C*, a careful examination of the roof and the floor of the stope showed that no sign of

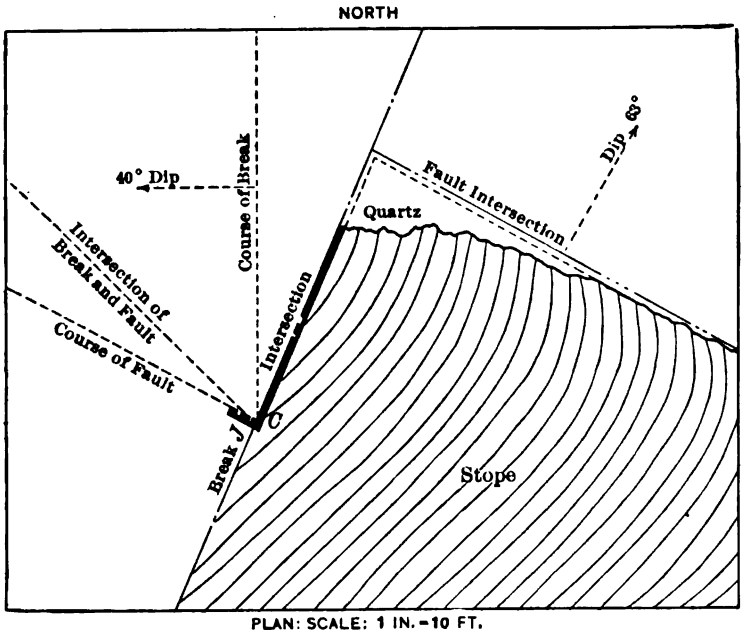

FIG. 4.—INTERSECTION OF FAULT AND BREAK, BERLIN MINE.

fissure existed in the foot-wall of the break. The fault-fissure at *C* is nearly enough in the plane of the fault, which cuts off on the north the next westerly stope, practically to identify it. On the hanging-wall of the break, within a few inches of the sharp intersection of the fault and break, are lines of motion parallel to the fault-plane.

The above conditions indicate that the movement of the north country was one movement, upon both planes at the same time, and therefore in the direction of the line of intersection of the two planes.

In Fig. 1A are shown the intersections, known and assumed,

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of the faults and breaks with the vein and with each other, and the entire hanging-country above these intersections is supposed to have been removed.

Figs. 1 and 1A were made about the middle of 1906, and represent the works and known or assumed intersections as of that date. One exception to this latter statement is, that the raise from the No. 8 level encountering ore at point *C* was begun and completed subsequent to the introduction of the lines of intersection.

Intersections of faults and breaks, where not surveyed and known, were then supposed to have the direction N. 45° W.; and the average angle of the intersections of the vein with the breaks was taken as N. 20° E.

Some careful estimates, made since the preparation of Figs. 1 and 1A, and involving all of the principal vein and fault intersections, including both north and south faults, show these intersections to have an average direction of N. 81° W.

A revised consideration of the breaks shows that the more important breaks have an average dip of 40° W. with a strike, as near as may be, N. and S.

Taking the average course of the vein as N. 45° E., and its average dip at 45° SE., we may, with the average surveyed and known intersections given above, and the known dip of the faults—viz., 63° from the horizontal—determine the average strike of the faults. This has been thus found to be N. 59° W.

The intersection of these planes of the average fault and break, as given above, strikes N. 43° W., instead of 45° as shown in Fig. 1, and dips in that direction 29.5° from the horizontal. The planes of the average fault and break, as given above, make angles with each other of 92° and 88°, the upper angle being 92°. The line bisecting the obtuse angle between these planes runs S. 12.5° E., and dips in that direction 57° from the horizontal. If these calculations be correct, this is the theoretical direction of the pressure or force which produced the two systems of fissures, here called breaks and faults. The existence, before erosion, of a rock-mass, known to be several hundred, possibly several thousand, feet in thickness would, through the weight of such a mass, supplemented by some lateral pressure, easily account for the fissuring; and gravity alone might be sufficient to explain the movement.

The miner, however, is more interested in the direction and

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extent of the movement, than in the question, just how it was produced.

In prospecting for the continuation of the Berlin vein, north of the north fault, the problem is complicated by the fact that the break-fissures found in either wall of any of the faults, do not necessarily correspond in their relation to each other with those in the other wall.

The same is also true of the faults. Thus, in the slab of andesite between the breaks *F* and *I* (see Fig. 2) are two known faults, and, in all probability, a third fault. At least two of these, if continuous, should be shown in the large open stopes on either side, but they are not to be found there.

It is also true that the north fault, is probably nowhere in this field a single fault, but that the total movement has been upon two or more nearly parallel fissures, with a slab, or several slabs, of rock, and a segment, or several segments, of vein between them. Moreover, the vein, as a rule, though not always, is uniform in direction and size; hence the identification of the opposite ends of a fault crossing any particular portion is not generally to be expected.

Three occurrences of the Berlin vein, beyond the most northerly known branch of the north faulting-fissures, are known.

At a point 800 ft. N. and 900 ft. W. of the top of the incline, but not shown in the drawings, is a segment of the vein, developed by a short tunnel and a shallow incline. This segment is cut off by a reverse-dipping fault, striking N. 48° E., and dipping about 80° southeasterly, developed by the innermost 100 ft. of the lower tunnel, which is driven upon it. The movement here has been in the opposite direction from other faulting movements shown, and the vein from which the segment at the surface has been cut off has not been found, but is still below the bottom of the lower tunnel.

There is also at the same place a break with reverse dip known to cut the vein, with a northwesterly course, and a dip of about 45° NE. This may be the southern edge of a series of reverse faults, with a reverse displacement of the vein.

Some work on the No. 8 level, east of point *C*, done since the preparation of Fig. 1, also indicates one or more faults with reverse dips and displacements; but these are as yet not well enough defined to be described here.

On No. 4 level, at the point marked (+ 40) in Fig. 1, is the

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south edge of a segment of ore which has been followed northeasterly and upward for a few feet. The total movement of the north country, which has here been upon three, possibly upon four, presumably parallel fissures, indicated by this occurrence, is, as nearly as can be determined, about 400 ft. in the direction N. 45° W., at an angle of about 30° from the horizontal. The total vertical component of the above movement is about 200 feet.

The vein at point (+ 40) was found by drifting along the most northerly branch of the north fault and breaking into its hanging-wall.

The third ore-occurrence, sought for by raise at *C*, and found since the intersection-lines in Fig. 1A were outlined, is at Station *C*, in the raise from No. 8 level, at which point the intersection of the vein with a break was found 55 ft. above the level. Here also a normal segment starts off. Although it has been temporarily interrupted further north, by a reverse-dipping fault, it is without doubt a continuation of the vein from the northwest corner of the most easterly segment of the mine.

Opposite this point is doubtless a double fault with an intervening segment, as shown in Fig. 1A.

This ore-occurrence indicates a total movement of the northwest country N. 45° W., and at an angle of depression of 33°, of 420 ft., and a total vertical drop of about 220 feet.

The position of the ore at point *C* was therefore 20 ft. further in the line of the movement and had 20 ft. more vertical displacement than is indicated at the ore-occurrence at (+ 40) on the No. 4 level. It was, however, the probable extent of the movement, as indicated by the ore found at No. 4 level, that suggested the raise from No. 8 level by which the vein was recovered.

There are exposed in the stopes of the Berlin mine, many instances of breaks and faults, which fade or run out to nothing, one of which, not shown in the drawings, but clearly enough in evidence in the mine, is found in the southwest corner of the second stope from the east end of the field. In this case the break, normal in its planes, intersections, and in all other respects, begins with a mere seam in the foot-wall of the stope, increases for 20 ft. or so, to a point where the vein-displacement is about 4 ft., then decreases for 20 ft., to a feather

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edge, leaving no noticeable fissure beyond its ends in the roof or floor of the stope.

One is here impressed with the fact that fissuring and faulting is the habit of the rock-mass. In this connection is perhaps worthy of illustration an observation, shown by sketch in Fig. 5, of the freshly uncovered side of the No. 4 level then being driven. A quartz-veinlet, about 1 in. in thickness, was in a few feet faulted normally three times, and by reverse faulting twice, between the roof and the floor of the level.

The bright, white quartz against the dark andesite told its story as though just from the press.

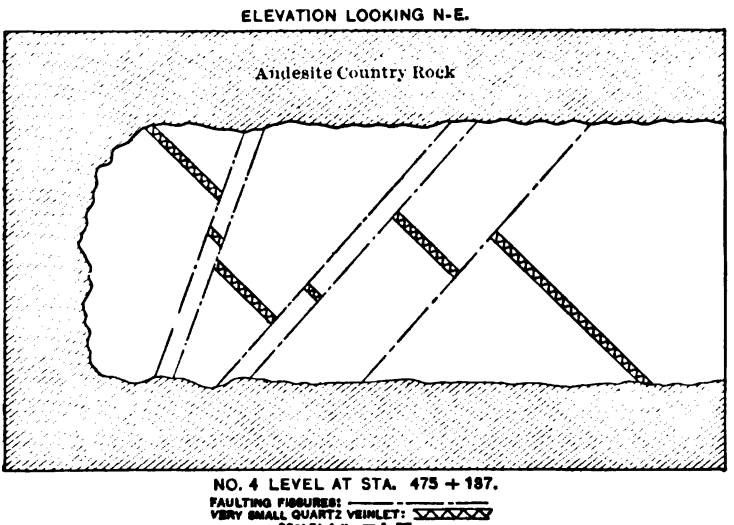

FIG. 5.—QUARTZ-VEINLET, SHOWING NORMAL AND REVERSE FAULTING.

In Figs. 1 and 1A an attempt is made to show pictorially the underground works and stopes, and in the two together the relations of the works on the Berlin vein and the breaks and faults which have disturbed it. In this illustration all that is really known of the matter has been found in the underground works shown in Fig. 1. The structure remote from the underground works is entirely assumption, based, as far as possible, upon the known ground as it existed in the middle of 1906.

From a practical standpoint, the extraordinarily disturbed condition of the rocks in the Berlin mine is very unfortunate. Without attempting to give details, it is evident that the prospecting and development of the vein in such a broken country must be unusually troublesome and expensive.

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Geology of the Exposed Treasure Lode, Mojave, California.

BY COURTENAY DE KALB, LOS ANGELES, CAL.

(New York Meeting, April, 1907.)

THE Exposed Treasure gold-mine has, for the past four years, been one of the largest producing mines of Southern California, its annual output having constituted 1 per cent. of the total gold and silver production of the entire State. At the present moment the property is idle, owing to the large quantity of water encountered on the lower levels, which will require the installation of a powerful pumping-plant before operations can be recommenced. Moreover, a prompt change in the character of the ore has occurred at water-level, which makes imperative an extensive campaign of development in the region of the unoxidized ores before a plant adapted to their treatment can be definitely decided upon.

The character of the changes encountered in these deeper ores makes the geology of this deposit a matter of importance, not only for the immediate district, but for the desert region of Southern California in general, where many mines exist having in the oxidized belt conditions that, in many respects, resemble those in the Exposed Treasure mine.

The deposit is situated in an apparently isolated butte about 2.5 miles south from the town of Mojave, on the Mojave desert. The butte, though apparently isolated, is in fact geologically part of an extinct volcano, known as Soledad butte, which rises out of the plain 1.5 miles SW. of the mine, to an altitude of 4,650 ft. above sea-level. Other buttes also rise from the desert plain toward the south and east, and again to the westward, all being closely related geologically to Soledad butte,—the whole constituting a single system as to origin and time. Since the end of the period of active volcanism in this region, there has been extensive denudation, the ancient plateau having been dissected during an epoch of apparently exces-

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sive precipitation. The plateau is known, through well-borings in the gravel-fill of the desert, to have been cut down to a depth of 1,600 ft. below the present general level of the desert, and a reconstruction of Soledad butte from the angle of rest for lapilli, deduced from remnants of the ancient ash-cone still remaining, shows that it may have towered to a height 2,500 ft. greater than it now possesses. Other evidences of great activity in denudation on the Exposed Treasure butte are quite in accord with this estimate for Soledad.

Fig. 1 is a map of a part of the Mojave desert and the Tehachapi mountains,—Soledad butte being shown near the center.

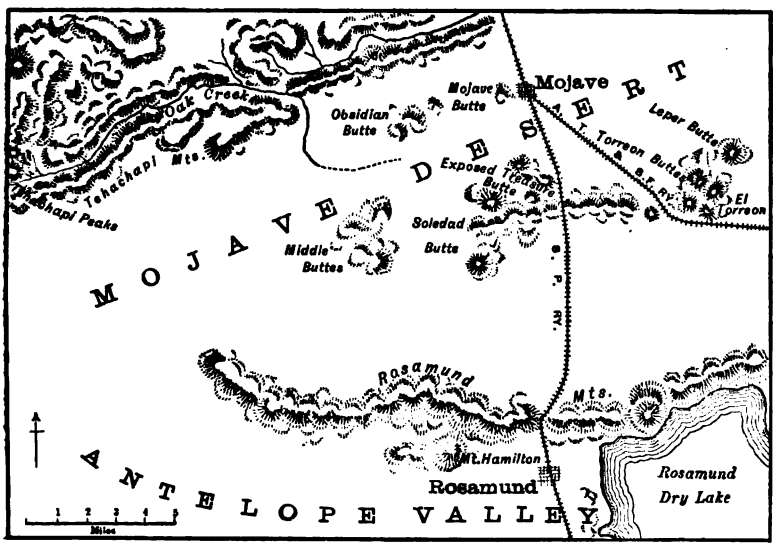

FIG. 1.—MAP OF A PORTION OF THE MOJAVE DESERT REGION, SOUTHERN CALIFORNIA.

The floor of the plateau consists of typical granite, extending to an unknown depth; and, being identical with the granite in the Tehachapi mountains, 5 miles to the northward, it is presumable that we have here the granites underlying the Tertiary sedimentaries, which still constitute the characteristic feature of the Tehachapi mountains in their eastward extension from the line of the Southern Pacific Railway, although all traces of such sedimentary rocks are wanting toward the west, where this range culminates in the tripartite Tehachapi peak, 8,052 ft. high.

The granite itself has been invaded by extensive pegmatite

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flows, determining the position of a low range of hills, which—for want of a name—I have for convenience called the Rosamund mountains. Pegmatite dikes also exist on the Exposed Treasure butte, and on the Torreon group of buttes, 5 miles to the eastward. As the Torreon group is connected with Soledad by a practically continuous chain of low hills, all lithologically related to Soledad, it may be fairly assumed that the original lines of weakness, contributing as one cause to the subsequent volcanic eruptions in the district, were those established by the pegmatite flows from the lower portions of the old granitic magma. It is worthy of note that these pegmatite dikes rarely show, by samples taken at random, a value in gold lower than 20c. per ton, and many of them, particularly in the Rosamund mountains, often assay as high as $1 per ton. The granite itself is never barren, but seldom carries more than 0.001 oz. of gold per ton. The absolutely universal dissemination of gold throughout all rocks in the entire district, requiring no refined methods of analysis to determine its presence, is a noteworthy circumstance.

The great mass of Soledad butte, as well as of its outlying hills, some of which were solfataras of the central volcano, consists of intensely acidic rhyolite-porphyry. Extensive fissuring has occurred in every direction, and all fissure-planes and zones have been further silicified, with abstraction of the alkaline feldspars, resulting in rocks often superficially resembling quartzite, sometimes possessing a porcelain-like texture, and a quality of resonance which has led to their being locally called phonolite. The fissuring has mostly occurred under slowly applied pressure, which has induced flowage of the porphyry, and even has caused it to become intrusive as dikes through the upper portions of the parent-rock. The flow-lines developed in the massive porphyry often give the rock the appearance of being contorted slates, while, on the other hand, the flow-dikes possess a granitoid structure which, on field-examination, would lead to their being presumptively identified as quartz-diorite. Microscopic investigation, however, proves that these dikes are only crushed rhyolite-porphyry, squeezed into crevices in the surrounding mass of porphyry and adjacent granite. No granite exists on Soledad at a higher elevation than 600 ft. above the desert-level, but it shoulders upon the

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neighboring buttes, and exists in isolated masses, these being remnants from the denudation that has almost obliterated all traces of the dikes of porphyry that must have extended upward through the uplifted granite.

In Fig. 2 the relation of the porphyry to the granite is clearly shown, the remnants at the points of deepest denudation on the mountain mass demonstrating the previous existence of

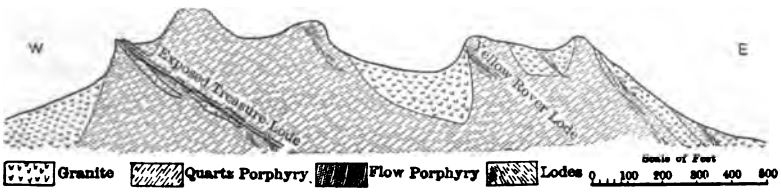

FIG. 2.—CROSS-SECTION OF EXPOSED TREASURE BUTTE.

large granite masses above. Fig. 3 is a sectional view showing the bottom of one of these wedges of granite in an adit tunnel on the Yellow Rover claim of the Exposed Treasure Mining Co. The fracturing of the lower point of the granite, and

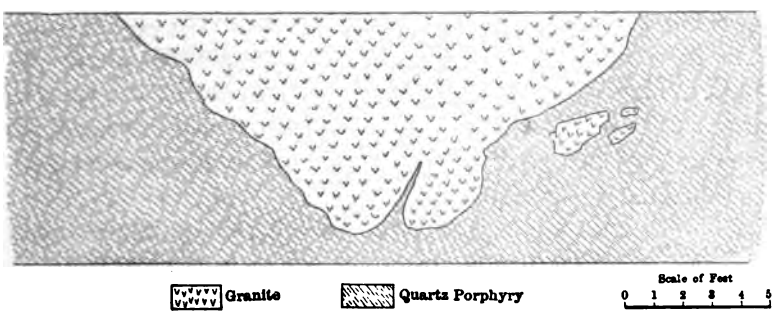

FIG. 3.—LOWER POINT OF GRANITE WEDGE UPLIFTED BY QUARTZ-PORPHYRY.

near-by inclusions of granite in the rhyolite, are particularly interesting. Noteworthy, also, is the fact that no contact metamorphism has occurred, the granite being almost as fresh in contact with the porphyry as within the granite masses themselves. The rhyolite-porphyry on the Exposed Treasure butte, and on nearly all the hills surrounding Soledad, has evidently flowed into its present position in a pasty condition, and at no greatly elevated temperature. At the south-

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eastern end of the Exposed Treasure butte, however, and at the volcanic stock constituting the Torreon, where solfataric phenomena were present, the porphyritic character of the eruptive disappears entirely; although in the center of the present porphyry dome on the Exposed Treasure butte the phenocrysts are splendidly developed, often attaining a major axis 1 in. in length, in a ground-mass which has undergone epidotization. In the mines in Soledad, where the porphyry has been revealed at considerable distances from the surface, the phenocrysts are not usually so well developed, and the ground-mass shows less alteration to epidote; it is also often quite fresh, and unchanged by the formation of secondary minerals, except in so far as silica has been introduced, as previously explained.

That Soledad was an active volcano is clearly proven by the important remnants of the ash-cone lying around the base of the mountain, notably abundant on the east side. There exists, moreover, on the west side of the mountain, a mass of volcanic tuff, buried under nearly 1,000 ft. of subsequent effusive rhyolite, the tuff having been compressed until it has developed horizontal cleavage, splitting the rock into layers of from 0.25 to 2 in. thick, as perfectly as the bedding-planes of a shale. This compressed tuff-remnant, 30 ft. in thickness, as revealed by denudation, makes it evident that there must have been at least two periods of volcanic outbursts connected with the effusion of the rhyolite alone.

The great acidic magma found its relief-vent chiefly at Soledad butte, but the uplift was general over a large area, and other vents existed at the Middle buttes, 4 miles west of Soledad; Mojave butte, now an inconspicuous hillock, where the rhyolite just emerges above the desert sands, 2.5 miles north of the Exposed Treasure butte; and Leper butte, a twin white shaft of quartz-porphyry standing solitary on the plain about 2 miles NE. from El Torreon,—the latter being the best type of volcanic stock in the district, though another fair example is found at the southeast end of the Exposed Treasure chain of buttes. The rhyolite also appears in the Rosamund mountains, and at Hamilton butte. My explorations in the Tehachapi mountains have failed to reveal any extension of the rhyolites into that range.

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Subsequent to the rhyolite eruptions there was an outflow of andesites through extensive fissures in the Tehachapi mountains, forming one great system of dikes across the eastern edge of the Tehachapi plateau, and another 3 miles ENE. of the great Tehachapi peak. This outflow also reached the surface at Obsidian butte in the desert, and at one point on the ENE. flank of Soledad butte.

Three distinct periods of faulting are traceable in the Exposed Treasure butte, the first being a series of clean-cut fractures, approximately S. 80° E., with a maximum horizontal displacement of over 20 ft., and a vertical displacement of about 5 ft., the fissuring being unaccompanied by crushing or brecciation. One effect of this faulting was to oppose porphyry against granitic faces, thus disturbing the original relations of the dikes and the intercalated granite masses.

The second movement produced extensive rupture under shearing strains, resulting in excessive crushing of wide zones, traversing the rhyolites and granites indiscriminately, nearly at right angles to the direction of the earlier fault-planes. There are two related systems of shear-zones, one consisting of 9 parallel zones, each the *locus* of a vein, comprising also the Exposed Treasure vein, having an average course nearly due north and south, and splitting up into numerous branches in a NNW. direction, where they run into the other set of shear-zones, likewise palmate at the western end, where they merge into the related fault-system in the center of the great rhyolite boss constituting the mass of the Exposed Treasure butte. The course of this second set of shear-zones is S. 60° E., leading it directly into the solfataric stock at the southeast end of the ridge, where it again splits into numerous finger-like branches. It is, moreover, less continuous than the major system of shear-zones, and was seemingly caused by resultants of the original force, which met with a resistance in the homogeneous Exposed Treasure boss of rhyolite, producing a complicated branching of fractures. The north and south zones, therefore, may be spoken of as primary, and the NW–SE. zone as secondary. Magnificent grooving, like the best examples of glacial grooving to be seen in the north, occurs in many places on the foot-wall of the Exposed Treasure vein, in one stope an area of more than 100 by 300 ft. being furrowed into parallel

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waves, some of which are 200 ft. long, 18 in. high, and approximately 3 ft. from crest to crest. These groovings bear N. 33° E., and dip 31° 15′ from the horizontal, while the dip of the foot-wall on which they occur is 34° 30′ N. 80° E. Closely corresponding evidences of the direction of movement are found throughout the mine, as deep as 800 ft. from the outcrop, and for over 1,000 ft. in length. The amount of throw or displacement has been measured with certainty at one point, from one original cross-fault, showing in the foot-wall, to its mate in the hanging-wall, revealing a total movement of the hanging-wall of 32 ft. toward the NNE. Similar evidences in the Yellow Rover and Boston mines, parallel with the Exposed Treasure mine on the east side, indicate that we here have shearing-zones accompanied by block-faulting on a large scale, the general movement being due to an approximately horizontal thrust coming from the direction of Soledad butte.

These parallel shear-zones, now converted into metalliferous lodes, with extensive chutes, and lenses of pay-ore, all dip toward the east, those which outcrop at a considerable elevation having a steeper angle of dip (even as much as 60°) for a certain distance, then flattening rather abruptly to inclinations varying between 32° and 38°, gradually growing flatter in depth until, in the lower workings of the Exposed Treasure mine, the dip is only 27°. Furthermore, at the same absolute elevations, these parallel veins maintain identical dips, so that the parallelism is almost perfect throughout.

After this fissuring of the region, extensive silicification occurred, apparently unaccompanied by replacement. It was evidently a mere cementation of the crushed zones with silica, probably extruded from the cooling rhyolite mass in a colloidal condition, resulting in masses and infinitely ramified veinlets of chert, along with which was consolidated much FeS_2 as cubic pyrite. Kernels of this chert and its included breccia—the latter now consisting of granite and again of rhyolite, depending upon whether the fissure at that point was traversing one or the other of these two formations—are found frequently in both the Exposed Treasure and the Yellow Rover mines, and they rarely contain as much as 0.02 oz. of gold and 0.05 oz. of silver per ton, with complete absence of copper. This is universal throughout the mines of this group.

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On practically all joint-planes throughout the unfaulted portion of the rhyolite boss (through the most highly porphyritic and unaltered portions equally with those where alteration has been profound), the same skeleton of cherty silica occurs, stained blackish-brown from iron oxides. On Soledad butte this extrusion of the overplus of magmatic silica is even more marked, but it contains a much smaller proportion of iron.

After this period of shear-faulting and subsequent cementation by silica, the veins or lodes were again subjected to faulting, in this case there being apparently no horizontal component, ordinary normal faults being produced. The effect was to re-brecciate the old cemented shear-zones. The formation of the metalliferous veins now commenced, the product being typical replacement-deposits. The silicates in the original breccia were, to a large extent, replaced by silica and metallic sulphides in the deeper portions, calcite becoming more abundant at higher levels until it finally became the predominant mineral, filling the interspaces between the cherty skeleton which had formed the cementing matrix of the earlier breccia. The calcite was of a liver-brown color, from mechanically contained manganese and iron compounds, and as the calcite in the upper portions of the veins dissolved away in advance of the denudation, the liberated manganese and iron oxides, together with clay, worked their way downward, so that in time great bodies of ore remained, consisting of a siliceous skeleton filled with a soft blackish-brown mixture of ferruginous clay and manganese dioxide, having much the appearance of an impure "bog" manganese. Throughout these masses were numerous blocks of the original cemented breccia, and the secondary breccia recemented with silica and calcite.

In the upper part of the veins, chrysocolla was a fairly common mineral, occurring both in the residual blocks of recemented breccia and lying detached in the soft manganiferous filling. It is also evident that the latest faulting had at places, temporarily at least, produced open fissures, as the occurrence of water-worn boulders, from the size of small pebbles up to 6 in. in diameter, would indicate. At one point in the Exposed Treasure vein, 40 ft. from the present outcrop, was a very remarkable mass of several tons of such surface *débris*,

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cemented by calcite, while smaller pockets of such gravel, and isolated boulders, are common everywhere near the outcrop.

As stated before, while the lodes are continuous, and often of great width, sometimes being 40 ft. and more from wall to wall, the pay-streaks, from 4 to 15 ft. in width, lie in well-defined chutes and overlapping sheets or lenses. It is noteworthy that only those chutes or lenses which now reach the surface contained important quantities of calcite and manganese dioxide. In the deeper-seated lenses, which had no direct connection with the outcropping upper lenses, the absence of the above-named minerals is conspicuous, the ore here being entirely siliceous, except for residual blocks of the original breccia cemented by chert. The processes of decay, however, have extended also to these deeper-seated masses, the alteration consisting in sericitization and kaolinization, the latter applying chiefly to remnants of the old granitic breccia. The result has been to produce a semi-friable mass, including kernels and blocks of all sizes of the harder unaltered ore. Chrysocolla is also fairly abundant, and copper carbonate occurs universally, often in large amounts. The remnants of the earlier chert-filling, while frequently heavy with pyrite, contain no copper, but the residual masses of the unaltered secondary quartz always contained chalcopyrite in considerable quantities, along with marcasite, galena, and sphalerite. These kernels also presented another interesting phenomenon, illustrative of the processes of decay still going on. They were always surrounded by the friable sericitized ore, becoming "honeycombed" nearer the kernel, the latter being discolored by large amounts of the green copper carbonate, and even copper sulphate. Near the outer portion of the harder mass the chalcopyrite had been either converted into bornite or coated with a film of this mineral. Within the kernel the chalcopyrite remained unaltered. It appeared that during protracted epochs of drought, to which the desert is subject, the moisture had been withdrawn from these kernels by the combined action of evaporation and capillarity, the copper sulphate in part reacting with the chalcopyrite to produce bornite, and in part either crystallizing out on evaporation, or becoming partly converted into the carbonate. It was also uniformly found that such un-

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altered copper-bearing kernels were richer in the precious metals than the altered friable ore. The altered ore bore manifest signs of extensive leaching, and where it had become almost completely decolorized by the removal of iron, the precious metal contents had nearly disappeared, and such ore never contained copper except in the form of chrysocolla.

The absence of sulphides in all the ores, except in the cherty skeletons, and in the undecomposed kernels of hard ore, was very complete. The mill-concentrates (150 into 1) had an average composition of SiO_2, 30; FeO, 37 (mostly from Fe_3O_4); and MnO_2, 12 per cent. These concentrates never contained more than 1.5 per cent. of sulphur.

In the lower friable siliceous ores, the ratio of gold to silver was as 1 to 12, while in the upper mangano-calcitic ores the ratio was as 1 to 72. Assays of gold-scale, and of coarse gold panned out, from all parts of the mine, showed a remarkably uniform alloy of 1 part of gold to 0.461 part of silver. The silver in the upper portion of the mine was present almost wholly in the form of silver chloride.

On the assumption, from the evidence, that the abundance of chlorides would prevent the leaching-out of the silver and its reconcentration below water-level, and that the ferric and cupric sulphates would have abstracted large quantities of the gold, which would be re-deposited lower down together with the copper in the form of secondary enrichments, it was natural to predict an ore below permanent water rich in these metals, and relatively lean in silver. It would be difficult to conceive a nicer justification of theory than that which was afforded when development at length extended below water-level. The ore consisted of a hard bluish-gray mass of original chert-cemented breccia, re-cemented by quartz, with partial replacement of the granite and quartz-porphyry by silica, heavily impregnated with sulphides, among which were considerable quantities of chalcopyrite, bornite, and some covellite. The gold-content of the ore had increased 150 per cent. above the average in the friable siliceous ores on the upper levels, and the ratio of the gold to silver was as 1 to 2.

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The Ore-Deposits of the Joplin Region, Missouri.*

BY F. L. CLERC, DENVER, COLO.

(New York Meeting, April, 1907.)

THE lead and zinc region of SW. Missouri is interesting, not only by reason of the value of its output, which ranges in the neighborhood of ten million dollars a year, but even more because of the facilities which it offers for the study of certain forms of ore-deposits, of which the *loci* and genesis are somewhat obscure. Although the mineral species found in the region are few and of common occurrence, crystallization has taken place on a generous scale, and unusual forms abound in many combinations, presenting examples of pseudomorphism, paragenesis, metasomatic replacement and the action of mineral-depositing and mineral-dissolving waters. For the mineralogist, lithologist, and specialist in certain lines of metamorphism, it has peculiar attractions, while the systematic geologist will perhaps find its chief claim to attention in the simplified problem it presents, in the study of ore-deposits. In addition to the phenomena above enumerated, it presents instances of well-developed comb-structure in the ores, fissures and faults, and slickensides—features usually associated with extensive movements in the earth's crust, and believed by some authorities to prove the deep-seated origin of the ores. In connection with these occurrences, the absence of all igneous rocks from this region is noteworthy and significant. For that reason, the simplified case presented by the Joplin region, from which these rocks have been eliminated as a factor, is specially valuable.

Having had unusual opportunities to watch the development of this district, I have been deeply interested in the bearing of the revelations of the Joplin mines upon questions of universal importance to the science of ore-deposits; and it was to my great regret that the project of a meeting of the American Institute of Mining Engineers at Joplin, proposed in 1892, was

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found to be impracticable. Such a meeting would have given to many experienced field-observers an opportunity to see for themselves many things which they cannot so clearly recognize or so decisively judge at second-hand through the descriptions and arguments presented in this paper.

The authority which Posepny has given to the theory of the deep-seated origin, and the deposition by ascending waters, of all sulphide ores, has without doubt enlarged our outlook, by constraining us to pause and review the evidence. His definition of "ascending waters," seems, however, in a final analysis, to reduce itself to this—namely, "those which deposit sulphides," and that of "deep underground circulation" to the water below the "zone of oxidation." His term "Barysphere" seems to imply, that because the average density of the earth has been proved to be more than double the density of any sections of it that are open to observation, therefore towards the center of the earth there must be accumulated a preponderance of the heavier metals. His words, in the translation, are "that is to say, the deep region is the peculiar home of the heavy metals."[1] While this surmise may be correct, I think few geologists or mining engineers will admit that it has much weight in the argument. Generalizations are useful only when they do not require the ignoring of essential facts. The paper of Chas. R. Keyes,[2] presented at the Mexican meeting of 1901, states the case strongly against such hasty generalizations.

The secondary enrichment of certain portions of an ore-body can often be observed in the development of a single mining district. The theory by which it has been explained[3] leaves little to be desired. It was a brilliant generalization, and I think one of the most important contributions, so far as immediate results are concerned, that geology has recently made to mining. Where this enrichment is confined, as it were, to a single set of apparatus, presented by natural conditions, its successive stages can be observed and proved. Such a migration of an ore-body, or of any of its constituents, may be likened to

[1] *Trans.*, xxiii., 247 (1893).

[2] Diverse Origins and Diverse Times of Formation of the Lead- and Zinc-Deposits of the Mississippi Valley, *Trans.*, xxxi., 603 to 611 (1901).

[3] *Trans.*, xxx., 27 to 177, 177 to 217, 424 to 448, in the contributions of Van Hise, Emmons, and Weed (1900).

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the movement of a whiff of steam, a wreath of smoke, or a flock of grasshoppers. The motion of the separate individual particles of which it is made up cannot be followed; but the resultant motion of the mass is evident, and the mass preserves definite outlines. The idea is so suggestive that it may lead us far afield, and I think there is danger in speculating over too wide a range. If we should call some observed concentration, the mth, and another, that could be proved to be later, the $(m+n)$th enrichment (bearing in mind always that it might be accompanied by x impoverishments elsewhere), it would be seen at once how indeterminate the problem may become. I shall make no application of this theory in the present paper, although there are cases observable in this district to which I think it will strictly apply.

Much has been written of the Joplin district. My statements are confined mainly to the area bounded on the north by Center creek, and on the south by Shoal creek, and especially to the high rolling prairie-lands between these streams; but I believe that many of my deductions are applicable to a much more extended area. Standing on one of these uplands, and looking towards either of the creeks, one sees that the ground slopes, at first gradually and then quite abruptly, to a nearly level bottom, through which the creeks wind, and that it rises beyond the creeks to the general level of the high prairies. These valleys are largely the work of erosion, and the bottoms are much wider than is required by the volume of the present streams, even in flood. There is evidence, in many places, of the filling of the bottoms by detrital material, and, in some places, of the raising of the bed of the creek-channel besides. Instead of a single bluff between the prairie and the bottom, there is often a strip of rocky and broken ground, sometimes several miles wide, consisting of ridges, intersected at frequent intervals by cross-valleys, which meet at the ridge-line and depress it, but discharge in opposite directions. These strips of rugged country are covered with a dense growth of black-jack oaks, and are marked "timbered" on the earlier Land Office maps. The exposures show generally chert and bright red clay; and where the strata are not horizontal, they dip, seemingly, in all directions. Erosion by

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surface-water and accumulations of local drift are much in evidence. It is difficult to trace any evidence of faults.

For a proper understanding of the problem to be discussed, it is necessary to state concisely the system under which mining has always been conducted here, and to indicate the unexpected difficulties, arising from this system, which must be encountered by any student of that problem.

As a rule, the owners of land in this district, known or supposed to contain mineral, do very little which can be called mining. They lay it off into mining-lots, or lease it to so-called mining companies, who then sub-divide it. The work carried on by these companies usually consists in sinking a number of pump-shafts, and in putting in and operating pumps. On the surface they furnish water to the washing-plants. By the terms of the mining-rights granted (that the lot shall be worked in a "miner-like" manner, I believe the phrase is), they have, but seldom exercise, an indirect control of underground work.

Custom has fixed the size of mining-lots at 200 ft. square—less than one acre each. They may be "registered on" by any individual or partnership acceptable to the company; and such registration secures, not a sub-lease, but a "mining-right." The distinction is important. The rights are forfeitable by failure to prosecute work, or to comply with the terms of the owner or company, and are granted for a fixed period. It is held that the ownership of the ore remains with the grantor. He has the right to say to whom it may be sold, and at what price; to have it weighed on his own scales; to collect the purchase-money, and to deduct his royalty, and (if he runs pumps) his "pump-rent"; and the remainder he pays over to the registered occupant of the lot, or his agent, as "the contract-price for work done." In practice, the rigor of these contracts is somewhat abated. The equitable claims of the miners are considered, in privileges of renewal, preference in registering on adjoining lots, and other ways. Most mining partnerships are unincorporated, but of late years a number of incorporated companies have taken formal mining-leases on several mining-lots, from the owner or original lessee, and have made special terms. The success of these corporations has not been conspicuous. Under the system by which the

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district has been developed, and which is still prevalent, the party mining the ore must first find it and then extract and prepare it for market. He sinks the shafts, runs the drifts, puts up hoisting-machinery, and crushes and cleans the ore, at his own cost. Notwithstanding these onerous conditions, many a comfortable fortune has been made by miners from a single lot.

The owner or the leasing company marks the lot lines on the surface, but does not take much trouble about underground lines. Miners on adjoining lots must settle these matters among themselves. Sometimes litigation results; but usually a tape-line is sufficient to adjust matters, though occasionally a surveyor is called in. Naturally, a miner who has found good ore is anxious to take out as much of it as possible, and therefore leaves the necessary supporting pillar on the next lot. It is etiquette, in visiting a mine, not to ask where the line is, and not to seek to know the course of the ore, or the point of the compass to which a drift is running. I have examined hundreds of working-places, often when I did not know the miners, and can recall only a single instance in which I was refused permission to make an examination.

The defects of this system are glaringly apparent; I have elsewhere[4] pointed out a few of the less obvious elements of strength and elasticity which it undoubtedly has exhibited. The defects involved the failure of the system to develop the mines thoroughly and systematically, and to preserve any record of what has been taken from them, and what has been left. This failure explains one of the many difficulties which attend the study of the district, and may excuse, to some extent, a certain vagueness in the report of some of my own observations—a vagueness which cannot be avoided, in cases in which no measurements were taken, or records kept, or when the evidence has been destroyed, and contemporary witnesses cannot be found. There is, in general, unquestionable danger that, because of the lack of records of old workings, the study of deposits now being mined may fail to disclose the relation which once existed between these deposits and those which were worked long ago, and that, from observations so limited and

[4] *Mineral Resources of the U. S.*, U. S. Geol. Survey, vol. i., pp. 368 to 373 (1883).

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widely separated, conclusions not sufficiently comprehensive may be drawn.

The theory here advanced to explain the origin and form of the lead- and zinc-deposits of the Joplin district, was first suggested by me in a little sketch of the district written in 1887, to encourage Mr. John N. Wilson in the publication of statistics which he had collected, covering the production of the region by companies and mining-districts, for the year. From his beginning can be traced the regular weekly publication of these statistics, which has come to be taken as a matter of course.

This theory may be summarized as follows:

1. The location of the principal ore-deposits of the Joplin district is due to a system of surface and underground water-channels, which was once much more closely connected with the surface-drainage than it is now.

2. The agency which diverted the surface-waters from these old courses was geological and not chemical.

3. This diversion has been effected within comparatively recent times, long since the close of the Carboniferous age.

4. The ores, as we now find them, were deposited after the surface-waters had been largely excluded by mineral-bearing-waters, which found access to these old channels, and a retarded passage through them.

5. This old system of surface and underground drainage is strictly analogous in form and origin to the present drainage-system, but entirely distinct from it.

6. Where the later system intersects the ancient system, surface-waters have rapidly cut out and dissolved the ores.

7. The ore-bodies of the region form a true system of connected and ramifying ore-veins, presenting as definite a problem for study as do the surface and underground streams of the present day.

In the lack of precise data, I do not expect to prove these theses rigorously; but if I succeed in making them appear probable, it may be hoped that the combined investigations of the U. S. and State geologists, with the co-operation of mine-owners and miners, will bring together a mass of evidence which will be decisive one way or the other. If the theory is not true, the sooner it is discredited the better for all parties

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concerned. If it is true, and if it be followed out to its logical conclusions, it may lead to important developments in the region.

Mr. Schmidt,[5] in his report of 1874, has noticed "that the principal ore-deposits have not been found on the larger streams, but on their smaller branches." Let us follow one of these smaller branches across the broken ground, beyond the point where it shows surface-wash and carries water all the year round, up and on to the high prairie ground which includes the true water-shed. After leaving all traces of surface-wash, shown in the dry gullies from which the soil has been removed, we are still able to follow a slight depression of the surface, through which the surface-water runs off. Following carefully one of these minor valley-lines, we notice similar slight depressions, coming into it from both sides. Following any one of these will bring us to the actual top of the high prairie, forming the divide between adjacent creeks. There we shall find a more or less level expanse, which appears to maintain a uniform height above the water in the larger streams. Drill-holes in this area will encounter a larger proportion of flint, and therefore a smaller proportion of limestone, in some places than in others, and will often go down many times the depth of the deepest shafts in the district, without striking ore-deposits or broken ground. This difference in the amount of flint may be due either to the persistence of local colonies of silica-secreting organisms, during the long periods in which the Mississippian series was being laid down, or to local deposits of siliceous materials in shallow waters, closely analogous to the sand-bars which we to-day find in muddy bays, the channels between which are filled with lighter limy silt. However they may have been formed, these cherty ribs are an important feature of this geological formation. They must have had from earliest times some effect in determining the lines of fracture in these strata, and some influence on the subsequent erosion. It is probable that this effect has continued down to the present day.

A shaft now sunk on the slope of one of these upland swales

[5] The Lead and Zinc Regions of Southwest Missouri, by Adolf Schmidt and Alexander Leonhard, *Report of Missouri Geological Survey*, pp. 381 to 502 (1873–4).

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will probably strike, below 20 ft. of soil, gravel, much altered chert and clay, a solid surface of limestone. On the top of this limestone, there is likely to be a thin film of water, scarcely enough to wet one's fingers in, but sufficient for the prairie "cray-fish," which bore to the surface from the shallow pools which collect in depressions, and sufficient also for domestic use. This is the true well-water (*l'eau phreatique*) of the upland prairies. The dip of the limestone will conform to the slope of the topography. If it is followed down by an incline drift, or by trial-shafts, it will be found to reach its lowest point nearly under the valley-line, at which point there will be one or more cracks, following the course of the valley. By drifting along the top of this limestone, in either direction, at right angles to the valley-line, another parallel break will be found, nearly under the point where the slope meets the plain above.

A topographic map on large scale would show that the valley lines have a dendritic arrangement, and that the smaller twigs of neighboring branches interlock. A simple rumpling across a line of stress, will not suffice to explain these observed surface synclinals. They suggest rather a marginal puckering around a central dome. It is not difficult to imagine a force, acting within the earth's crust, which might have this effect; but in view of the short wave-length—that is, the small distance from crest to crest—of this contorted limestone, such an explanation is not satisfactory. Another shaft, sunk at a point beyond which none of these lateral branches can be traced, may very likely show a very slight saucer-like depression, scarcely visible at first sight. A shaft so located is almost certain to be on a "water-hole." Instead of solid limestone, it encounters broken limestone, in the form of fragments, much decomposed on the surface, and, between them, crevices filled with clay. When chert is found under these fragments it usually retains its position in the stratification, but is broken through transversely, so that it can be taken out without blasting. Thinner strata of limestone, between flint layers, are not only broken into fragments, but are actually perforated by water dropping through the flint strata. From a single shaft, I have seen half a dozen or more of these perforated masses, some of them too large to go into the hoisting-tub, sent to the surface by hooking the rope through the holes. Here we have evidence of the

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excavating work of underground water, and the suggestion that it may have brought about the subsidence of the surface-rocks to produce the synclinal troughs above observed. Without doubt, this work of dissolution and abrasion, acting for a long time, would remove enough material from the lower strata to cause the upper strata to settle down. The fewest number of fractures which would enable a solid ledge of rock to accommodate itself to this lower level, would correspond to the three lines of break already observed. The first point to be made clear in this connection, is that the material removed has not been taken from any one channel; it has been leached or washed out from all parts of the mass accessible to the water within this limited water-shed. The chief line of attack of this water has varied from time to time, and its effect was measured by its volume and its dissolving-efficiency. It is not probable that a cavity existed at any time, at all corresponding to the amount of material removed.

This operation of weathering throughout a mass, either above or below ground, can be observed in the shrinkage of an old mine-dump. Originally it had sharp outlines, and was composed of pieces of different size, hardness, and solubility; but after a few years' exposure, the track is twisted out of shape, the general height is diminished, lighter materials have been carried away, and even the harder have begun to crumble. After one generation, only its broken and scattered skeleton may remain to mark its site. A still better illustration, perhaps, is the melting of a snow-pile between street and sidewalk, where the ice-sheets are the last to disappear.

In addition to the solid material dissolved or carried away in suspension by these underground waters, there are actual flows of soft mud from a higher to a lower level, which are too commonly observed in this region to permit doubt of their effect in producing local subsidence of the surface.

Upon the reasonable assumption that one underground stream, traced as above described, is representative of others, there is such a stream corresponding to each of the branches of the wet-weather surface-flows; and these underground streams constitute a system closely related to the surface-system.

A roof requires a certain thickness to support itself. In following one of these underground streams, a point is reached

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where it forms a true cave, either because the roof has become self-supporting, or because the stream finds an open course through the ridge which divides it from its neighbor. On following it still further, evidence may be found that the roof has fallen in; or, in other cases, that a surface-stream has cut into the underground channel. In either event, accumulations of much mingled *débris* will have resulted; and it is often not easy to distinguish between the work done by underground and by surface-streams, respectively.

One further point must be noticed in passing—namely, the effect of open cavities in preserving the ridges and higher ground from disintegration and weathering, by rapidly draining off from them the surface and dissolving waters. They have also a similar effect in protecting the ores in the higher ground.

In following a little further the course of these streams I shall call them the "shadow-streams" of the surface drainage-system, not because I think this is a name which should be retained, but because it will help me, in the absence of a diagram, to picture actual conditions. The shadow of an object does not have the same form as the object itself; its lines are shortened or lengthened, thrown to one side or the other, or even reversed, according to the position and form of the surface which receives the shadow. The shadow-streams are thrown on whatever surface is impervious enough to carry off their water. If this surface rises or falls, or inclines to either side, the shadow-streams will be correspondingly distorted. It should be noted, that while a surface-stream or run-off has normally four directions of possible movement—forward, downward, to the right and to the left, an underground stream, if running full, has two additional directions—upward and backward. If a tree represents the surface-streams, we must choose, to represent the underground streams, a tree on which the branches join the stem at all angles. Evidence of the existence of these shadow-streams (some of which had a continuous flow before pumping became general) is found in all the camps of the Joplin district. In some cases it is possible to explore the caves through which they run, to study their configuration and its causes.

The shadow-streams of the present surface-drainage do not,

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in general, contain ore-deposits. They are interesting by reason of their striking similarity in form (suggesting a similarity of origin) to an older system of shadow-streams, connected with an older surface-drainage, which does contain mineral deposits.

The regimen of rivers, streams, and mountain-torrents, and their agency in shaping topography, has been much studied; an elaborate classification has been made; and an extensive vocabulary has been invented or adapted to express their present and past stages. The work of underground streams might be similarly described. But as the amateur trout-fisherman does not need to know anything of this classification or vocabulary in order to make a shrewd guess, several hundred yards before he comes to it, on which side of the stream the next deep pool will lie, or where he will find the head, and where the tail, of a riffle, so the mining engineer, following one of these underground streams, by noticing its rise and fall, the material cut through, and the inclination of the strata, can foresee on which side will lie the accumulation of *débris*, and on which the open channel. Moreover, as the fisherman, so long as the trout rise, and he does not meet with "No Trespass" signs, does not worry about the continuation of the stream, and would say, if asked, "It probably starts small somewhere, and eventually reaches the ocean;" so the miner, as long as he finds ore in these channels, and keeps on his own property, does not worry about the continuity of the channel.

I explored one of these caves on the Hopkins farm on the north side of Turkey creek, four miles east of Joplin. It had been stopped up for several years, and I hired a man to dig it open. His shaft was sunk at a point he had marked about a third of the way down the slope of the hill on which the house stands. It struck the cave at a point where it makes a right-angled turn to the west. I followed the longer branch in a northerly direction, and came, a little way in, to a clear stream of running water, a foot or more wide and an inch or two deep. For some distance, the roof was generally flat, and high enough to permit me to walk erect. The nearly vertical left wall was made up of horizontal layers of alternate lime and flint. The right wall was concealed by a thick bank of tallow-clay, which sloped from the level of the roof on the right down to a nearly

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vertical cut, made by the stream in its foot. The clay had the consistency of lard, and had retained with absolute fidelity the impressions made upon its surface by the tools, clothes and fingers of the men who had visited it several years previously—showing that the water had not risen much, and that the drip from the roof was slight. The stream flowed over a bed of the same clay, and some of our steps sank into what appeared to be cracks in a rock floor.

The cave made numerous and short turns; and I noticed horizontal cracks, formed by the spreading apart of two layers of chert, extending under the roof and into the right wall, almost continuously, as far as we were able to penetrate. These gaping cracks seemed to narrow as they passed in; but our lights could not penetrate to the end of them. In places the cave was filled almost to the roof with fallen slabs of chert. Some of these still hung at dangerous angles, attached to the roof near the right wall, and it was rather a ticklish matter to crawl through the small openings left, and not brush against them. We passed through several of these openings where, of course, we lost the stream, and came to wider places where we found it again. In one or two other places the stream disappeared below the floor; but it reappeared again and continued to within 40 ft. of the end of the practicable opening.

I squirmed along like a lizard for the last 30 ft., hoping to find an enlargement beyond. At this point the flat opening under the roof extended across the width of the cave, into the walls on both sides, and ahead farther than I could see. When I twisted around to crawl out, my candle was extinguished; my matches were wet; and I could rely only on the sentry whom I had posted to guard his light at the point where I had left the stream. In spite of my shouted caution, he began to trim his wick and thus put his candle out. Either his matches or his clothes were wet; and he could not get another light. However, we had only to find and follow the stream, and it would take us to daylight. So one of us was posted where we lost the stream, and the other crawled up and down until he found it again. In this way we worked our way out in the course of two hours, wet through and plastered from head to foot with tallow-clay.

After procuring dry matches, I explored the west branch,

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which was short. Near the end I found that the stream dropped into a lower chamber, which extended back under the floor of the larger cave. The dimensions of this chamber were about 6 by 8 by 3 ft. Dropping down into it, I saw that the stream ran into a hole in its floor. The probable junction of this stream with Turkey creek is a spring about a quarter-mile from this point.

My examination of this cave was not as thorough as I had hoped to make it; but it was satisfactory to a certain extent, in that no trace of mineral (unless the tallow-clay was zinkiferous) was found and I saw no incrustations of any sort. The fractures of both limestone and flint appeared fresh and but little corroded. In every other respect, so far as I could see, the cavity was completely like many which I had found filled with ore. The horizontal openings between flint layers in the right wall and at the end, if they had been filled with zinc-blende or blende-bearing material, would have been the counterparts of the "hard sheet-ground" elsewhere mined. The point worthy of particular attention is, that if these gaping openings had been encountered in a shaft, only a few feet away, the existence of the cave might not have been suspected, and it might have been difficult to explain how they had been produced. I can only estimate, in a general way, that the length of the cave explored was about 200 yd. (The popular estimate is several times greater.) But, judging from the topography of the country, I have no doubt that this cave has direct connection with an underground stream which I struck in a shaft more than half a mile to the northeast.

I could mention many other caves similar to this one, with water flowing through them, and all barren; but I do not know of any that have been surveyed, and I cannot give a more definite description of any of them. I will cite only two; and these can only be called caves upon the assumption that a number of horizontal openings between flint layers are the equivalent of a cave, and form, in fact, a section of its length.

The first is under the natural cut of which the Missouri Pacific and the Kansas City, Fort Scott and Memphis railroads take advantage, in going from Joplin to Webb City. Here an underground stream has largely done the grading for the railroads. In 1892, I sank a shaft a few feet east of the railroad right-of-way, and was obliged to put in, at the depth of 18 ft.,

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a pump, which was 12 in. in diameter with 6-ft. stroke. When it began pumping the working-barrel projected above the surface of the ground. A second pump of the same size was soon required, and the two had to be run continuously, night and day, at 22 stokes and upwards a minute, to hold the water down. The ground was broken flint and clay, and the water, which came always from near the bottom, was usually clear and potable, but filled with almost invisible spangles of bisulphide of iron (probably marcasite), which danced in the tank like motes in a sunbeam, and gilded with "fool's gold," for a quarter of a mile beyond, the bottom of the ditch that carried off the water. A little blende, in small crystals, was also found. When I last visited Joplin, seven years later, there were several productive mines working on both sides of this draw.

The other underground stream which I would mention was struck by a drill-hole near the center of the tract owned by the Center Creek Mining Co., at Webb City. This stream was so strong that it swept away all the cuttings of the drill. The water did not rise to the surface or (apparently) get to the main pump-shaft, only 200 yd. away. The mine-water of the district is too acid for use in boilers; but this stream, which passes among the mines but not through them, furnishes water of satisfactory quality, which has been substituted in boilers for city water, with a considerable economy.

A long chapter might be written on the mine-waters of this district, and the incrustations in the old workings. Unfortunately, I can cite no analyses, and do not know that any have been made. I saw the column of a force-pump, 10 in. in diameter, which had been taken out of a shaft at Webb City, and which was almost closed by a deposit looking like a mixture of clay and iron-rust, though the iron of the pipe was apparently not much affected. The pump had been running continuously, but slowly. The opening left was only 3 in. in diameter. On the other hand, in some of the mines of the district, a shut-down for a single week means that the mine-rails will be eaten through in the web, and picks and shovels ruined, and that a new clack-valve, if of iron, will not last long enough to unwater the mine. This sudden corrosion, which does not occur while pumping is continuous, shows the results of exposure to air and the action of oxidizing waters. Perhaps such phenomena ex-

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plain in part the legendary "Kobolds" of the German miners —malicious spirits, who work in the dark, hurry away exposed treasure, to be hidden and sealed up in safer vaults, and leave in its place worthless trash; who lose no opportunity to steal the miner's tools, and damage his property as much as possible. What is more important, this rapid oxidation may represent the first step in the process of secondary enrichment.

A system of underground water has been compared to a city's water-works, including the water-shed, storage-reservoirs, filter-beds, mains, laterals, house-connections, sewers and sewage-disposal works. For more complete analogy, we might interpolate two additional items—namely, the connection of a water-supply with steam-boilers by means of injectors (representing local solfataric and fumarole action in nature), and the connection with laboratories and chemical factories, for the production of comparatively pure mineral substances. It is to this last diversion of the water-current (which, in the case of a city, would probably amount to only a fraction of 1 per cent. of the total consumption, and, in the case here considered, will not be relatively larger) that particular attention is here directed.

It must be remembered, however, that in the underground water-system of nature there are no valves to be opened and shut. The water is always running. If the reservoirs are not emptied, it is because water is flowing into them at least nearly as fast as it flows out. Dams may give way; mains may cave in or be washed out; smaller conduits may be choked or otherwise thrown out of use; new connections may be made; and the level of waters in the reservoirs may rise or fall. The flow is always subject to hydraulic and hydrostatic laws. Efficient head, cross-section and friction control it. I cannot conceive that capillarity has any directive force when the capillary tubes are filled with liquid and both orifices are wet. Under these conditions it is simply a retarding force, like friction, which can diminish the velocity of the flow of a liquid, but, if time or area be varied in reciprocal proportion, cannot prevent ultimate movement. I find no difficulty in the way of water penetrating into cavities already full of water, provided there is a chance for an equal amount of water to escape, against a lower pressure, in another direction. Permeability is relative. Small leaks may be neglected in considering large flows; but this

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does not prove that the leaks have no effect. A rock containing but a few hundredths or thousandths of 1 per cent. of hygroscopic moisture may yield a highly concentrated solution, if this water be displaced by slow filtration. Chemical activities, known and unknown, play their part; but I have always thought that water-deposited minerals have been placed where we find them by vein-currents of water which, by reason of their differentiation from the circulation in the adjacent barren country-rock, have become more fully saturated with mineral salts.

Whether such currents be "ascending" or "descending," appears to me a question of relatively small importance. If this small fraction, either of the annual rainfall or of the artesian supply, which can act in a certain district, circulates slowly through beds containing even traces of vein-minerals, and finds its way into any open cavity, such as an old, disused, and partially choked system of underground water-courses, I think the conditions are furnished for just such deposits as are found in the Joplin district. If, on the other hand, it discharged into flowing streams, there would be dissipation, instead of concentration, of its mineral contents.

Mr. Schmidt, in his report of 1874, has described certain residual clays which can be identified throughout this region; and his explanation of them, as formed *in situ* by the undissolved residues of limestones and cherts, is not likely to be questioned. The tallow-clays, which are often zinkiferous, and are generally found associated with oxidized zinc-ores, have been analyzed, and were probably formed near where they are now found. The clay called "gumbo" is generally found in shallow basins, and perhaps never exceeds 10 or 15 ft. in thickness. It is generally mottled, shows ocher-yellow, purple and white colors, and looks much like the coarser kinds of Castile soap. When a miner encounters it in sinking a shaft, he discards pick and shovel, and cuts it out with mattock or axe. It closely resembles in appearance the gumbo which lies a short distance above the upper coal-bed of Pittsburg, Kans. The black shale (which I have elsewhere called "slate," that being the name given to it by the coal-miners when it occurs over coal) is found under widely differing conditions. Sometimes, as a layer only a few inches thick, it covers a comparatively large area, and, under these conditions, is often

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bleached to light ash-color and is very friable. Often it is found very solid, in unstratified deposits, the vertical thickness of which may exceed any of their other dimensions. A thickness of 100 ft. or more has been observed in the Webb City-Carterville district; in the mines around Joplin; at Galena, Kans.; and at Aurora, Mo. In such formations the shale requires heavy charges of dynamite to shatter it. Its laminations are distinct, but have no definite relations to the numerous transverse joints, which often break the shale up into rhombohedral pieces. A cross-section of one of these pieces, on the cleavage-planes, often shows within the rhomboid an oval of unaltered shale with a border of bleached shale. With this form of shale occur curiously distorted crystals of blende, from the size of a mustard-seed to that of a horse-chestnut; minute scales of white mica; crystals of dolomite; and nodules and crystals of marcasite. Numerous fragments of coal are often found with their partings of bone and thin seams of iron sulphide; as are also large bodies of what I have called "bottom-clay." Sometimes, if not always, these materials are as devoid of definite arrangement as if they had been dumped into a shaft from cars or wheelbarrows. Evidence of considerable movement throughout this heterogeneous mass is furnished by polished surfaces, dipping at all angles with the horizon, and a soft black gouge is evidently formed by the grinding together of the pieces. I have drilled through 40 ft. of this material, under more than 100 ft. of horizontal strata of alternate limestone and flint, where the upper ground was so little altered that the cuttings showed white up to the point where the shale was reached. I have also met with it at a depth of 150 ft.; but in that instance I was able to trace the crevice through which it extended almost to the surface. Similar occurrences have been reported by others. In the once rich mines on Sucker flat, I saw at the side of a shale chimney (that is to say, against a rib of solid ground) a beautiful expanse of slickensided surface, which I estimated to cover an area of 200 sq. ft. Unfortunately, the shale which formed the body of the mass, crumbled too easily to be detached from the rib. The only hand-specimen obtained, so far as I know, showed a beautiful mosaic, of ebony (shale), ivory (chert), and tortoise-shell (zinc-blende), finely polished and vertically grooved.

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The "bottom-clays" are quite hard, and distinctly gritty to the touch, and show little if any cleavage. They sometimes occur in considerable masses in the body of the black shale; and they likewise contain small crystals of lead, zinc and iron sulphides. Their appearance is the same as that of the material composing the "horses" or "hog-backs" which cut through the coal-beds at Pittsburg, Kans. If fragments of coal occurred only in connection with shales and bottom-clays, they would call for little separate study, but they are also found under widely different conditions. I have myself taken out of a pocket of ochery tallow-clay about a bushel of lump coal, the lumps of which were so completely plastered with clay that it required hard scrubbing to show what they were, and expose their little-weathered surface. These lumps lay in the shaft, with as little order of arrangement as if they had been dumped from a cart into a deep mud-puddle. Perhaps the most interesting form of the shale is that which is found in connection with "spar" ground, and in dark-colored deposits of minerals, in flint ground. The miners call it "selvage," but as it occurs throughout the ore-body, and not especially against the walls, that term does not accurately describe it. It can often be traced continuously to a bed of less-altered shale. The prevailing color of mineral-deposits in spar ground—that is, among dolomite fragments, or against a dolomite bar—is dark; and this black "gouge" is invariably associated with such deposits. Those deposits of ore which occur in flint ground, with no limestone in evidence, and show the greatest amounts of "secondary" quartzite as cementing material, are generally dark-colored, and contain more or less of this black "gouge." On the other hand, where the ore is all free—that is, where there is no cementing material—both the blende and flint are usually light-colored. There is considerable evidence to support the view that the dolomitization of the non-magnesian limestones of these Mississippi series is extremely local; and, without doubting that silica has been dissolved out of the cherts and siliceous limestones throughout the region, I believe that the decomposition of the shale was the most probable source both of magnesia for dolomitization, and also of the soluble silica ("secondary quartzite") which is often the cementing material in the ore-bodies. I have never seen an analysis of the

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shale or the bottom-clay, and I do not know how easily they are decomposed; but their close association with the most important ore-bodies so far discovered in this region makes them worthy of special study as one of its peculiar features.

These materials—the black shales, fragments of coal, bottom-clays and "gumbos"—I call the "drift" of the district. They are strangers in their present location, though they may have come only as far as from the next township. Geologically considered, their age dates from their arrival here. All clastic rocks are made up of the *débris* of older formations. This drift may have formed part of a true Carboniferous coal-measure, or even of an earlier formation; but if it was brought here in late Quarternary times it is Quarternary. The problem of such drifti s more than a local one. It reaches to the Rocky Mountains on the one side and to the Gulf on the other. How are we to explain the occurrence of the unsorted and highly heterogeneous material which we often find in beds of great thickness? In particular, how are we to account for these lumps of very friable coal—not coal in process of formation, but completely formed coal with its "bone" and partings? Such lumps must once have formed part of a regularly deposited coal-bed, consolidated in place, under a sufficient weight of roof. It is impossible to imagine how they can have been formed otherwise. They must therefore have been eroded and transported subsequently.

The action of a river, in the formation of a delta, the erosion of banks, and the formation and shifting of bars and channel, by which it is able to shape and occupy a wide valley, is well understood. Also the terminal fans of torrential streams, and the overlapping lateral half-fans of rivers subject to great rises, are generally recognized. In these cases, the motion of the heavier material transported is a combination of rolling and sliding, and the pieces show rounded or rounding forms. But there is another motion to be considered, which I may call "crowding." Who, that has been in a painfully struggling crowd, has not noticed how the crowd melts away at its edges, and how individuals, big and little, strong and weak, young and old, emerge with surprisingly little personal damage? The strong protect the weak, the large the small; and often family parties are not separated. It is on record that, in the Johnstown flood, a pile

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of china plates and a "mogul" engine were found in close proximity, and that the engine was more damaged than the dinner-plates. It can be seen annually how a great river breaks through not only artificial dikes, but also the barriers it has itself built at its normal stage. Some such "crowding" flood must have excavated and carried this drift-material.

Throughout this region, both in the ore-bearing and in the barren areas, blocks covering many square yards of ground, and 50 ft. or more in thickness, have been thrown down or up, as the case may be, and are now bounded by fractures transverse to the bedding of the rocks. These fractures answer precisely the definition of fissures, the faults produced by them have their throw and their hade, and their result, in producing dislocation of the strata, can be worked out by "fault-rules." But if they have been produced by the fall of the roof of a cavity, and are therefore limited in depth by the bottom of the cave, or if they are due to the settling of the upper rocks upon a lower stratum, the thickness of which has been reduced by the removal of a part of its material by water, we certainly cannot regard them as deep-seated fissures. In reply to a geologist, who showed me an ore-filled, faulted "fissure-vein" in the ore-body of one of Webb City's largest mines, I first pointed out how the "throw" of the fault diminished from the top to the bottom of the exposed face, and then, in a specimen of brecciated vein-filling, I traced with my finger a detached fragment of flint which would accurately fit against a larger neighboring piece if it were replaced.

The term "gash-vein," which some writers have used for deposits like these, has always seemed to me inadequate, being neither distinctive nor descriptive, and it has not met with general acceptance as the name of a class of ore-deposits. A gash is a gaping cut, presumably not deep. If it has any significance as applied to veins, it might be applied to those incomplete cuts (fissures), which are formed at the same time as a "true fissure," by a strain which is relieved as soon as the fissure is made. We often see such cracks around the funnel of caved ground. Or a "true fissure" may end in such arrested tears. Similar cracks, having no throw at their closed ends, are found alongside of fault-fissures.

I must also demur to the term "blanket-vein" as applied

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to any deposit in this region. A blanket in its proper place is a component part, and forms a distinct layer in the making, of a bed. A blanket may, it is true, be much torn and bedraggled, and be trampled into the mud of a wet camping-place, and thus suggest a vivid word-picture, serving, as it doubtless has done at Leadville, to show how widely the ore-bodies there differ from the idea of the ore-veins which was in the minds of the framers of the U. S. mining laws.

The term "sheet" is, perhaps, not equally objectionable, since, in its extended meaning, it refers to the form, rather than to the position or function as a covering, and (as in the expressions, "a sheet of paper," or "sheet-metal") usually implies a limited extent.

I do not doubt that extensive and deep-reaching faults will be found throughout southwest Missouri and the adjoining territory. Nor do I underestimate the importance of studying them, and the effect which the "litho-clases," which must have accompanied them, have had in developing the underground streams. I have studied only the shallow faults and fissures which are exposed in the mines, and by the work of the underground streams. The study of the larger, structural faults presents some difficulties, due to the prevalence of these minor faults, to local variations in composition of the rocks as originally laid down, and particularly to the weathering and cementing processes which have long been active. Mr. Wm. Wallace, in his special study of the lead-deposits of Alston Moor,[6] presents a beautiful example of what a detailed study of a complex system of faults and fissures is likely to reveal, and how far it may go towards explaining present topography, the location of the ore-bodies, and their probable origin.

I will not attempt to prove, as a general proposition, that the spaces now filled with masses of drift-material and ore-bodies are the same in form and mode of origin as those which are occupied by underground streams of the present drainage. That question must be decided for each case upon evidence taken on the ground. There are many deposits in which the evidence is not decisive either way; but if the Institute had visited Joplin in 1892, I could have taken the members to selected mines

[6] *The Laws which Regulate the Deposition of Lead Ore in Veins; Illustrated by an Examination of the Geological Structure of the Mining Districts of Alston Moor* (1861).

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which in my opinion illustrated every stage of formation shown by present drainage-channels. And however much difference of opinion there might have been as to the cause, there could have been little as to the sequence of events. The paper of Mr. C. E. Siebenthal on the structural features of the Joplin district[7] is the only expert testimony I have seen which bears on the subject. His four sections on the Cornfield tract cover a length of 800 ft. on the mineral belt, which, worked at intervals, extends southeasterly from the Nevada ground, north of Webb City, to the township line, a distance of over 6 miles. I have not the slighest doubt that, if these deposits were owned by one company, they would be worked and described as a single ore-body.

In this connection I may say, that the description of the Bleyberg lead-vein, given by Phillips,[8] could be taken as a remarkably accurate description of this ore-body, if necessary changes were made as to dimensions, directions, and geological formations, together with some allowance for individual impressions in assigning causes. This similarity, however, by no means proves a similar origin. Two further factors, due to local conditions, must be considered: the effect of geological structure on the form of caves; and the effect of the strong tendency to crystallize, exhibited by the minerals of this district, upon the resultant phenomena in the Joplin region. We must at once divest ourselves of all pre-conceived notions of caves, in limestone, as arched and domed chambers, connected by passageways, and exhibiting stalactites, stalagmites, and other forms of beauty, and realize what a cave must be in a region of alternate limestone and chert. In measuring the thickness of the chert-beds, we must bear in mind that they are made up of separate layers, seldom over 8 in. thick; that these layers usually have little cohesion, the one with the other; that they are very brittle and have been severely strained, even where not broken transversely. The consequence is that, if there is a cavity below, they will "ravel out" indefinitely. To illustrate, if it be assumed that the interstitial spaces in a pile of flint blocks dropped one upon another constitute 35 per cent. of its volume, the solution of 35 per cent. of the volume of

[7] *Economic Geology*, vol. i., p. 119 (1905–6).

[8] *A Treatise on Ore-Deposits*, p. 253 (1884).

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the solid rock might extend the "cavity" indefinitely in all directions, yet keep it always filled with broken chert. The enlargement of the cavity may be at the top or bottom or at either side, wherever the material was most soluble or the water most active. As soon as the roof broke down, or the sides were sufficiently undercut, there would be vertical or lateral compression, forcing the mass into a smaller space; and this reduction of volume would be indicated by corresponding surface-depression.

The *rôle* elsewhere played by dikes and igneous intrusions in controlling and directing the flow of underground water, is here taken by bodies of shale or clay, which are less permeable than anything else found in the region, and remain so, after yielding to any stress they sustain. The miners recognize this, by calling anything which cuts out their ore a "bar." This is a confusion of ideas which makes a description of a mine very difficult. I suggested in 1887 that these shales might be the proximate source of the minerals, and thus be the equivalents of the igneous rocks, in a part sometimes assigned to the latter. It is evident that these patches of shale were once much more extensive than we find them to-day. This is, however, of minor importance. I also suggested the close of the Ice age as the time when they reached their present position. This point also is not vital, and can only be judged in the light of further studies of the physiography of this and adjoining territory.

In other mining regions, the material which goes to the dressing-works is called "ore." Here, that term is reserved, in general parlance, for the cleaned ore as it goes to the smelters. The richness of Joplin "ores" is proverbial; but it is generally overlooked that these ores are really concentrates, and that the material sent from the mines to the crusher averages less in zinc than any other zinc-ore in the country which can be profitably exploited.

This anomaly cannot be explained by superiority of methods or machinery. These, it is true, are admirably adapted to local conditions, but the local conditions themselves, involving, as they do, intermittent operations on a small scale, and frequent removal of plants (which are consequently cheap constructions), are all inconsistent with close saving or economical operation. The explanation of the surprisingly favorable

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results in the Joplin district is found in the fortunate circumstance that the minerals of the ore have crystallized separately. There are lead and iron enough in all of the camps to reduce greatly the value of the zinc-ore if they could not be easily removed by jigging. I think too much attention has been given to the location of reducing agents and not enough to the force which determines crystallization, in producing workable ore-deposits at any particular *locus*.

The Vein-System of the Standard Mine, Bodie, Cal.

BY R. GILMAN BROWN, LONDON, ENGLAND.

(New York Meeting, April, 1907.)

I. Introduction.

Mines are interesting by reason of what they have done for man, or of what has been done for them by nature. Not all are interesting on both scores. Many profitable mines are commonplace to the geologist; and many presenting unique geological conditions have been sad failures commercially.

The Standard group at Bodie has produced in 25 years, $14,500,000, paying $5,000,000 in dividends.[1] On the other hand, this output has been mined (almost wholly above the 1,000-ft. level) from a system of more than 100 veins, ranging in thickness from 0.5 in. of "specimen rock" to 30 ft. of clay and quartz; distributed through a zone about 2,000 ft. in width, and representing from three to five distinct periods of formation. This group, therefore, by reason of its industrial record as well as its geological features, may fairly be regarded as doubly interesting.

The Bodie district occupies an island of recent (Tertiary) hornblende-andesite, surrounded by a complex of igneous rocks and breccias. It lies at the summit of the eastern foot-hills of the Sierra Nevada mountains, and within the Great Basin, a

[1] Regular mining began practically in 1879. Down to 1884, the gross product was $7,662,000, and the dividends amounted to $3,525,000. For the 10 years, including 1894, the record was: gross product, $1,996,000; dividends, $186,000; and for the next ten years, gross product, $3,101,000; dividends, $449,000.

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few miles from its western edge, marked here by Mono lake on the line of the Great Fault. The average elevation of the outcroppings of the veins may be taken at about 8,700 ft. The andesite projects from south to north in a great tongue, 12,000 ft. long and 5,000 ft. wide, reaching an extreme height at the northern end of perhaps 1,000 ft. above the adjoining small canyons. These separate it from the main mass of the foot-hill complex on the west, the north and the east, but the south end of the tongue is rooted into the complex. Surface indications point to a structural line of separation, very probably a line of fault, at the point of union. At the very heart of this andesite mass lies the Standard group of mines, the richest and most productive of the district. The original Standard mine consisted of but two claims; but during the past 10 years it has come to include the properties formerly known as the Bodie, Mono and Lent Shaft, on the south; the Bulwer and Belvidere, on the west; the Bodie Tunnel and Tioga, on the north; and the Summit and Harrington Tunnel on the east—in all about 200 acres. Portions of the surface are covered with large blocks of flinty quartz, of low value; there is also much *débris*, characterized by large feldspar (albite) crystals. I have observed a single straight face on one of these crystals 4 in. long. This remarkable occurrence is found as well in the upper workings, where the well-defined albite crystals are associated with quartz, and, in more than one specimen, the crystals are mere shells, studded internally with fine prisms of quartz. I have never identified gold in association with the feldspar, but have many times panned out gold from specimens of mixed quartz and feldspar. My impression is that the gold occurs only in the quartz. Some features point to the albite as a secondary constituent of the veins. But it cannot easily be identified deep in the mine, and the matter has not received much study, so that no definite conclusions have been reached concerning it.

In the main, the soil is thin; but along two lines of depression crossing the property, there is a heavy covering of what is locally known as blue clay. These zones, mentioned later in this paper, are probably due to surface decomposition of the andesite along lines of shattering. The surface is devoid of trees; but it is interesting to note that a section of silicified

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wood, part of the trunk of a tree that must have been 14 in. in diameter, was found in the wash below the Bulwer shaft.

II. The Vein-System.

This is made up of at least three series: in the order of their age, the Fortuna, the Incline and the Burgess; in the order of their yield, the Incline, the Fortuna and the Burgess.

Fig. 1 is a generalized sketch in cross-section of these series with their modifying dikes and faults. The oldest series,

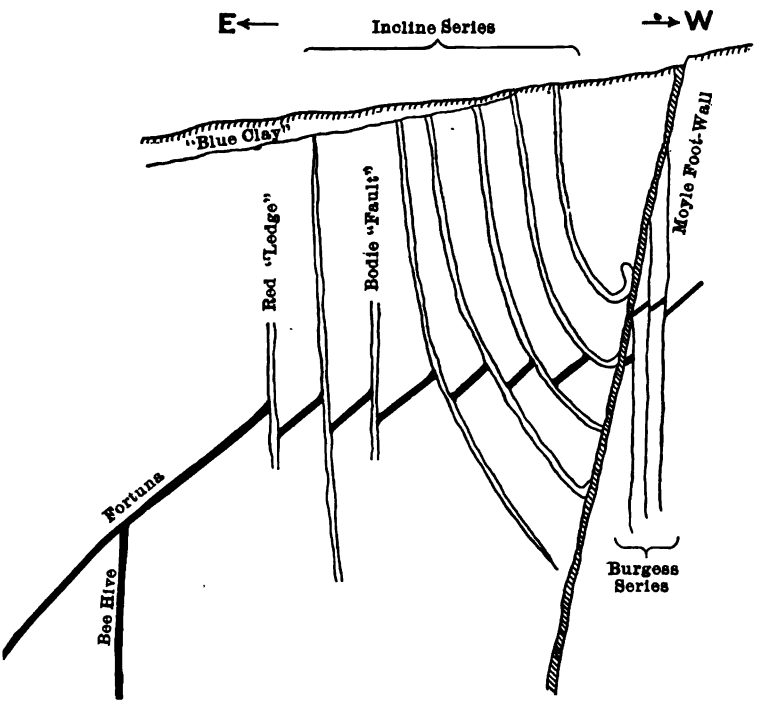

Fig. 1.—Cross-Section of the Vein-System of the Standard Mine.

the Fortuna, is dislocated by each of the succeeding ones, so that its veins are cut up into short segments. The resulting complexity is increased by the divergence of strike, which makes of each disjointed vein-segment a polygon that is far from a parallelogram. The other two systems are, in the main, on the hanging-wall and foot-wall sides, respectively, of a strong line of fault—the Moyle "foot-wall"—which cuts boldly through the mines. A few unimportant members of the

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Incline series are found to the west of this fault, but none of the Burgess veins are found to the east of it, unless, as is only barely possible, the enriching-veins, of which mention is made later, belong to this series. In addition, there are at least two dikes faulting the Fortuna, the Red Ledge and the Bodie "fault." Between these and the Incline series, no intersections have been developed which would determine their relative age. They are several hundred feet from the Burgess series, and probably do not cut any of its members.

The ore of the Fortuna is characterized by hard, flinty, at times bluish, quartz, "frozen" on to the walls. The bullion produced has the largest percentage of silver of any of the series; the visible gold being noticeably light in color. The Incline series shows a relatively small proportion of coarse gold and stands lowest in silver percentage; and the Burgess is characterized by exceedingly coarse gold, 50 per cent. of which will stay in the mortars by its mere weight, without the aid of inside amalgamation. In both the Incline and the Burgess series, the quartz is well banded and sharply separated from the walls.

We have thus three sets of veins occurring in close juxtaposition in the same country-rock, and at times intersecting one another, yet each with a marked peculiarity of gold occurrence, and each so distinct in the appearance of its quartz that hand-specimens can be recognized with ease. Added to this is the diversity of dip and strike, making it as difficult to ascribe the fissuring in all cases to the same set of forces, as the gold to the same source.

1. *The Fortuna Series.*

The general characteristic of this series is a strong fissure, at times accompanied by andesitic breccia, on one or the other wall, or partly silicified in the vein, as if the mineralization had followed the line of an earlier dike. This, however, is not completely demonstrated; nor is the breccia by any means a constant feature. Only two veins of this series have been productive (though a third, unimportant one exists)—namely, the Fortuna and the Beehive, which yielded the richest ore of the old Bodie mine. The later vein-series and dikes have all "had their fling" at the Fortuna; and the consequent dislocations have complicated mining operations so that notwithstanding the richness of the veins, superintendent and foreman must

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have been kept in continued uncertainty and anxiety; and it is small wonder that more than one of the broken segments was overlooked in the first exploitation, and left to reward the systematic explorations of later date. At one place on the 430-ft. level of the Bodie, where both the later series cross the Fortuna, the latter is cut three times in the same plane in the same cross-cut. It is easy to imagine the excitement that must have reigned when what probably appeared to be three parallel veins, all of great richness, were exposed at short intervals in this one cross-cut.

As will be seen in Fig. 1, the Fortuna is the only vein with marked easterly dip. It is also the only one presenting the characteristics of a strong, deep-reaching zone of fissuring. Unfortunately, this permanency is of structure alone, and does not extend to the contents; for in the lower portions the vein rapidly becomes impoverished; sphalerite, from being almost absent, increases to a large percentage; gold values weaken and almost die out; and silver diminishes to a few sparse patches of wire in masses of impure, bluish white kaolin.

Near the lower boundary of the "bonanza" zone, a vertical vein, the Beehive (Fig. 1), characterized by extraordinary silver-contents, drops from the Fortuna. Only a few inches in width and cased in hard rock, it was worked with profit some 150 ft. down. In the bottom it became poor and dwindled to a mere "marker." Here we have a notable occurrence: a vein carrying good gold-values (the bullion ran from $9 to $10 per oz.) is robbed of all its gold, and much of its silver, by a "dropper" characterized principally by silver, which itself soon "peters out," as regards both fissure and contents—while the original vein continues down into the earth as a well-defined fissure, marked by extensive decomposition of the walls (a feature noticeably absent above, where flat stopes now stand open over large areas, with no sign of weight on the few scattered old timbers).

The Fortuna has been explored below the bonanza zone by at least three levels and their connecting raises, and has proved barren for a considerably greater distance on the dip than was productive above.

But the problem presented by the Fortuna is not yet fully stated. The bonanza region of this vein was most strongly de-

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veloped and richest where the veins of the other series intersect it, though rich ore was found not alone on these lines, and it does not appear to have been essential that the bodies in the later series should extend down to the Fortuna. To state the situation otherwise, the Fortuna has always yielded fine ore at zones of intersection with the two junior series, when those series carried ore-bodies above, whether the junior did or did not extend to the horizon of the Fortuna. Rich zones have been found on the Fortuna equally up and down the dip from the intersections. It should be added that in all cases which I have seen, the Fortuna retained its characteristic flint-like frozen quartz, and the junior veins their characteristic banded, more porous quartz. At the same time, rich ore has been found in the Fortuna 100 ft. or more from any vein-crossings, as, for instance, in the zone above the Beehive, east of any profitable veins of the Incline series.

This generalization regarding ore-occurrence was acted upon to good advantage soon after the consolidation of the Bodie with the Standard. The Fortuna had been considered unprofitable, and, indeed, had scarcely been recognized within the Standard territory. On the south line it showed as a most unpromising seam of white quartz. Having in mind that its strike would bring it under ore-bodies then being worked in the Burgess and Maguire veins, a drift was started which, in due time, opened up profitable stopes.

A final feature of interest in this vein is to be found in the internal crossings, or fractures, occurring within the body of the vein itself, and not extending far, if at all, into the walls. In most cases these represent lines of enrichment on one side or the other; if the stope is looking well one of these crossings is likely to "queer" it. From Fig. 2 a fair general idea can be gained of the appearance of a face in a Fortuna stope.

The above observations indicate that if there was secondary enrichment resulting from the intersections of the veins, it could not have been from the senior to the junior veins. That there was an enrichment seems probable, from the association of the ore-bodies; and it is not unreasonable to suppose that, since the bonanzas in the Incline and Burgess series were in the oxidized zone, a transfer of metal-contents from the surface would be likely to occur, passing downward into the underly-

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ing Fortuna, and that the currents would be likely to follow the lines of fracture in that vein now showing in the crossings, particularly as the general physical character of the vein is "tight." It seems probable, therefore, that the Fortuna was originally a silver-vein, with values derived from below through the Beehive, and that the extinction of gold-contents below the Beehive is merely a coincidence of the location of that vein at the lower line of secondary enrichment, which in turn may have been determined by the ground-water level, to-day not far below this point. This presents a fair working-theory, and one that in the main "holds water," though it gives no ex-

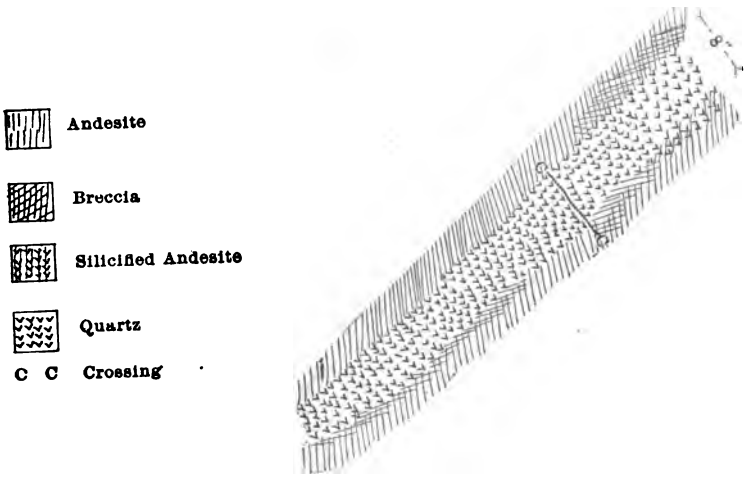

FIG. 2.—FACE IN A FORTUNA STOPE.

planation of the thick mass of decomposed rock disclosed by the deeper workings on the Fortuna vein.

The profitable zone of the Fortuna is not more than 1,000 ft. long; to the northwest the vein has never been identified with certainty north of the Standard shaft, though there is in the territory of the New Standard a nearly vertical vein of similar appearance from which ore has been taken in small amount. To the southeast it has been followed far beyond the Mono shaft into a region of decomposed andesite, indicated on the surface by the gulch and depression in the ridge between the Mono shaft and that of the old Champion.

One can hardly leave this subject of the much-faulted Fortuna without pointing out the seeds of legal complication which

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have here failed to germinate. Had erosion proceeded a little further, and had unhappy "A" located on an Incline vein (Fig. 1) and equally unhappy "B" located on the Fortuna just to the west, and yet more unhappy "C" located on the Moyle foot-wall, it is hard to see where conflicts of title could have stopped among the entangled extralateral rights and artificial end-lines. As a matter of fact, I know not how, under the conditions that actually obtained, litigation was escaped. The Fortuna apex had not been identified, but six or eight narrow

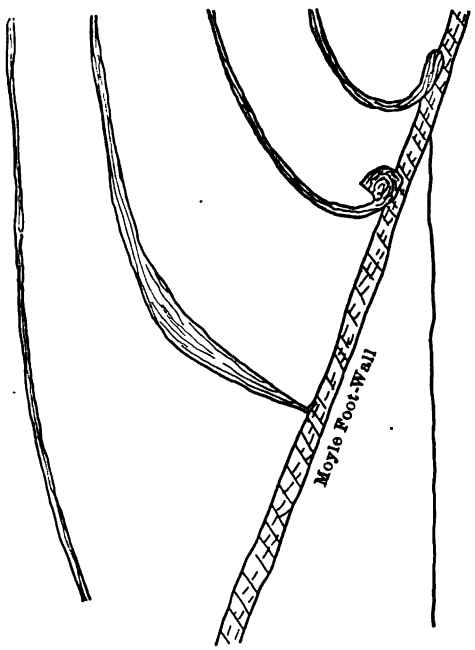

FIG. 3.—CROSS-SECTION OF THE VEINS OF THE INCLINE SERIES.

locations were made over it, on the veins of the junior series. It was a rare piece of good fortune for the mine and the camp that these were merged into the property of the Bodie Consolidated Mining Co.

2. *The Incline Series.*

An idealized cross-section, Fig. 3, shows the striking structural features of the veins of this system. More than once they have been classed as gash-veins, in distinction from fissures, and it may be that this is the case, though the evidence is not

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all that way. Their marked characteristic is the fading-out of values at about the 500-ft. horizon, but I do not know of a single instance where the continuation of the vein has been disproved, while in one case, at least, that of the Gildea, the vein is found in normal width and appearance below the Fortuna, at some 700 ft. depth. As we now see them, many of this series have no chance to reach depth, being cut off by the great fault-line, called the Moyle foot-wall. There is nothing to show the extent of motion on this fault, but while it has apparently been large, since the zone of comminution is thick, running from 10 to 40 ft., and many of the veins are curled up on it in great masses of broken quartz (Fig. 3), evidence from the Fortuna region, to be given later, is contradictory. As an argument for the gash-vein hypothesis, it should be noted that there are no veins below this fault corresponding to the Incline series above.

There is some evidence that there are at least two groups of different age included in the Incline series. This is found in the marked dissimilarity of the vein-filling. One set is characterized by massive, beautifully banded flint-like quartz, at times in a firm vein and at times badly cracked, but always associated with quantities of red ocherous clay, such as would be derived from wall-attrition in feldspathic rock. Fine sharp chips of the same quartz can be washed from the mass, and there is little doubt that the values this stuff carries come from these chips. In these veins rarely are both walls well defined. The Main Standard and Incline veins are instances of this group. The other is characterized by less clay and by well banded, rather porous quartz, noticeably soft and at times crumbly, as if crushed by weight rather than attrition. I have come across no intersection that would decide the question of contemporaneity, but in some instances the two sets are close neighbors, as is the case with the Incline and the Bullion. Here it seems unlikely that had the Bullion existed at the time the Incline was grinding on its walls, it could have escaped. The probability clearly is that the Incline series is composed of two groups, if not three. Accepting this, it is to be noted that in the later group there is a sparse scattering of black oxide of manganese; in one vein indeed it was such a marked presence that the name Black Ledge came as an inevitable consequence.

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I have said that the values in these veins fail at about the 500-ft. horizon. This is the lowest level at which veins of this series have been profitable, and the general average would be 100 ft. higher. This unfortunate fact has been largely offset by the multiplicity of veins, and the regularity with which ore-shoots parallel each other in different veins in the pay-zone. Many times has this recognized condition been the basis of profitable development—*e. g.*, the discovery of an important ore-body by a cross-cut, run on the 318-ft. level to cut a vein that showed nothing of value on the level below, merely because parallel veins had turned out well.

In veins of the later group of this series it is difficult to determine the value of the ore by inspection, though gold shows readily in the horn. Indeed, it was for a long time obscure to me what caused the sudden fall in values, often occurring in raise and stope in their upward course. Many times it happened that we got into old workings and found the top of the stope with a good face of fine-looking quartz that would assay but a dollar or so; and it was only by the closest inspection of the physical conditions that the question was answered and incidentally many thousand dollars more won from the mine. In all of these cases small veins or "enrichers" have been found branching off, generally into the foot-wall, and carrying very high values. These were followed, some hundred feet or more at times, each yielding perhaps half-a-ton of high-grade rock per man per month. The presence of these "enrichers" once established, it was comparatively easy to identify them in the poorer mass of the vein below. Fig. 4 shows a typical instance of what I have described, and clearly indicates that veins that were very low grade at first have been enriched by the later formation of small high-grade seams in their mass. That these are not confined to the older veins, but branch out and up into the country-rock, argues that they are veritable later veins, and not merely segregations by surface-leaching from the original contents of the larger veins. Their smallness and high grade would seem to point to their belonging to the Burgess series, which remains to be described. But, having some of the Burgess characteristics they lack others —namely, the coarseness of the gold and the character of the quartz. Moreover, no noticeable faulting is accomplished by

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them; so that I rather think they constitute a third group of the Incline series.

It must not be understood that all the veins of the second group of the Incline series owe their chief value to these "enriching" seams; that they do not there is abundant evidence.

All the veins of the Incline series dip west. The typical width is from 14 to 20 in. Their walls are well defined and good. In their neighborhood the country andesite is thor-

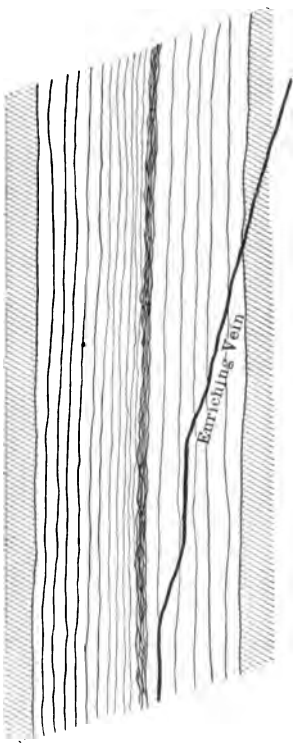

FIG. 4.—TYPICAL ENRICHING VEIN.

oughly oxidized, unlike the andesite around the Fortuna, which, even at high levels and when much decomposed, shows no iron stain and exhibits bright crystals of pyrite. A few unimportant individuals of this series lie below the Moyle footwall. The albite crystals alluded to were found near the surface in veins of this series. Another mineral peculiarity was a local occurrence of nearly pure crystalline calcite, filling the full width of the Bullion vein, on the 260-ft. level.

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3. *The Burgess Series.*

If we exclude the "enriching" veins from this category, as seems proper, all of these veins lie below the Moyle foot-wall. They are small, nearly vertical veins, characterized by a slightly banded structure, at times comb-like on an axial line and frequently with a rich scale of hard quartz frozen on one of the walls. This is so important a feature that many of the old stopes have been cleaned out and the scale removed by "pop shots" and moyle-and-hammer work at good profit. There is also at times a remarkable occurrence of "shale" on one of the walls that is an unfailing indication of rich rock. What this is, I am not sure: it has the appearance of a dried clay, mixed with sand, and shows in films, say one-sixteenth inch thick, suggesting an attrition-product. The wall-rock of these veins is noticeably less oxidized than that of the preceding series—naturally so, when it is remembered that few of these reach the surface, so that down-flowing waters are diverted into the more permeable fractured zone of the Moyle foot-wall. So rich have been some of these seams that more than once has a width of 0.5 in. been worth stoping. On the other hand, though the veins are characteristically narrow, at times values have been found some distance out in the country-rock. This was notably the case with the Burgess bonanza, discovered on the Bodie 300-ft. level, south of the Standard line. The late John Broderick, for years foreman of the Bodie, told me that when this was discovered the miners were preparing to discard the "porphyry" as waste, when he detected in it a generous sprinkling of coarse gold. This bonanza was mined for some 12 feet in width, and its location, only some 30 feet above the Fortuna, makes it probable that its richness was due to secondary enrichment from upper portions of the Burgess, according to the hypothesis already proposed regarding the enrichment of the Fortuna.

Not infrequently, angular pieces of andesite are found entirely surrounded by the vein quartz. The andesitic breccia, which has been mentioned as found at times in one of the walls of the Fortuna, is here also in evidence, and sometimes contains chips of a close-grained, coal-black rock, which does not occur in massive form elsewhere in the mine or neighborhood. No microscopic determination has been made of it, but macro-

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scopically it would be considered a basic trap. It has clearly been picked up from some distant rocks by the viscous breccia.

The veins of the Burgess series have not been profitable for so great a distance along the strike as those of the Incline. Their "pay" has been included between east and west planes some 600 ft. apart, as against 3,000 ft. for the Incline. About 1,000 ft. on the Fortuna has been profitable.

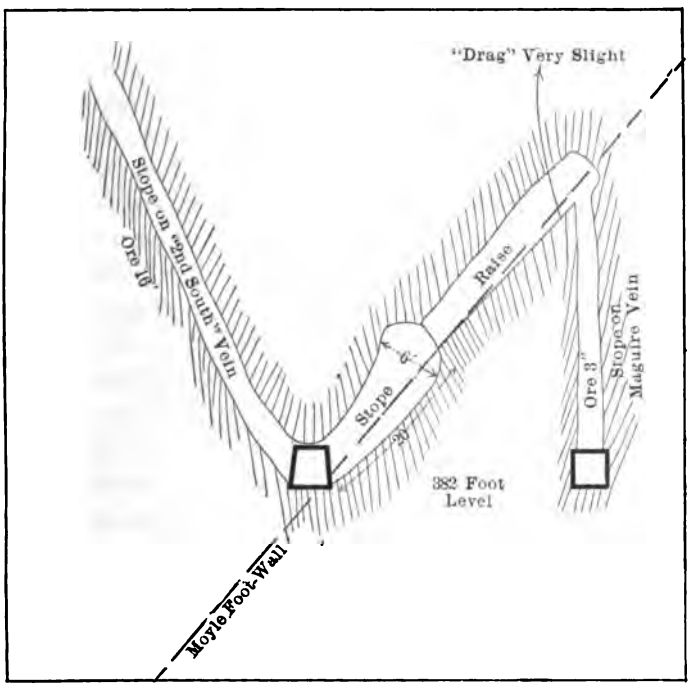

FIG. 5.—EFFECT OF THE MOYLE FOOT-WALL UNDERGROUND.

III. THE MOYLE FOOT-WALL.

Little remains to be said of this. It shows now as a scarp from 10 to 15 ft. high, west of the Standard shaft, where the ground has settled on to the big stopes. This scarp extends for 1,500 ft., and is a striking feature of the topography. Underground, the action of the fault has been to roll up the ends of the veins it cuts into big arches of crushed ore. (See Fig. 5.) To the north it dies out in the dead zone, but what becomes of it to the south is still a matter of speculation, as its intersection with the Fortuna, which it must fault, has not been identified.

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I rather think that this obscuration may be because it has followed the line of some other vein in crossing the Fortuna. If that is the case, the dislocation has not been great; and even allowing a gyratory motion on the fault—and some facts point to this—it cannot have been an extensive motion. This point has already been touched upon in the description of the Incline series.

IV. The Dead Zones.

There are two dead zones lying east and west across the andesite ridge, one to the south of Mono shaft, and the other some 2,500 ft. north and 800 or 900 ft. north of the Standard shaft. Both of these show on the surface as depressions and gulches on the westerly slope. Underground, they are characterized by heavy decomposition of the country-rock, and within their area no ore has been found, and most of the veins deteriorate to mere discolored seams. In the southern one, exploration on the Fortuna has been pressed for a long distance, yielding in places fine crystals of gypsum, evidently a secondary product. In this connection it should be remarked that in the Gildea vein small plates of the same have been found. It is probable that these zones have been determined by fracture lines, along which the rock has been brecciated. The absence of quartz-fragments argues that these fractures antedate the Fortuna, and the fact that the Fortuna and other fissures weakly enter the zone is what would be expected, for no fissure or line of circulation could be conceived as staying open in a pulpy mass such as this is to-day. Beyond the northern one of these zones, the veins come in again of good value, but in transit they have lost their identity and generally reversed their dip; the other characteristics are unaltered.

V. Dikes.

No careful study has been made of these, since they do not seem to have affected the mineralization. The principal one is the "Red Ledge," a breccia too decomposed for easy determination. Where it cuts the Fortuna it is about 4 ft. wide, and stained red. Assays of from 40 to 60 cents gold per ton are common from it. No intersection with veins of the other series has been developed; hence its true chronological position cannot be assigned.

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VI. Air-Cracks.

Open air-courses in the country-rock are common, particularly in the northern end of the territory; these may show a slight faulting of the veins. They have been found up to 6 in. in width, with an air-current strong enough to extinguish a candle, and have been sufficiently frequent to assist effectively in ventilation. The deepest one of which I have record was some 300 ft. from the surface, east-and-west in course and nearly vertical. They represent the last of the fissuring-periods by which the ridge has been visited, but it is not safe to assign to them off-hand the same deep-seated causes. Where they are especially prominent the ridge is nearly 1,000 ft. above the canyon, and precipitous, so that they may easily be due to frost and the settling of the rock towards the free face of the end of the ridge.

VII. Résumé.

In this interesting vein-system, we have within an area 1,000 ft. by 4,000 ft. eight or nine successive sets of fissures, five of which are ore-bearing. The Fortuna series, the oldest, is much faulted, and probably was originally a silver-vein, and owes its chief gold-contents to secondary enrichment from the faulting veins. The Incline series comprises three groups, part of the value of the second coming from the small enriching seams of the third. The Burgess series is still later. Beside these there are the great fault of the Moyle wall, the dikes, the dead zones and the air-cracks, most of which are so interlocked and crossed as to give definite evidence as to their relative ages.

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Barite Associated with Iron-Ore in Pinar del Rio Province, Cuba.

BY CHARLES CATLETT, STAUNTON, VA.

(New York Meeting, April, 1907.)

AN examination of the census reports[1] for 1880, which contain a large number of complete analyses of typical American iron-ores, indicates that the existence of barium sulphate in intimate association with iron-ore is rather unusual. Of 94 analyses reported there, but four show any barium sulphate, and the maximum content amounts to 3.04 per cent. The note of the occurrence of considerable quantities of barite in connection with certain iron-ores found in the Pinar del Rio Province of Cuba may be of interest.

These deposits are found near the little village of Francisco, about 25 miles west of the city of Pinar del Rio on the Western Railroad of Havana. The principal ridges are anticlines, and consist of very thick deposits of sandy shales and slates containing a large number of little veins of white quartz, which, by the gradual disintegration of the mass of the material, is left covering the surface. This quartz occasionally shows sulphide, but is generally barren of any such indication. At times lenticular masses of a more or less pure limestone were observed in these slates. The shales, which are of light color, are occasionally talcose.

The formation would tentatively be classed as the Lower Cambrian, and the heavy ridges of comparatively horizontal limestone found in the anticlinal valleys as the Upper Cambrian.

Along the crest of these slaty ridges deposits of iron-ore are found which are apparently in place, and were probably formed by replacement of masses of limestone similar to those noted elsewhere. The ready disintegration of the slates, and the

[1] *Report on the Mining Industries of the United States*, Tenth Census of the United States, vol. xv., pp. 583 to 601 (1880).

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fact that the only product of such disintegration which holds together in masses of any size is the iron-ore, has left the ore exposed to an unusual degree, and permitted its transportation over long distances and its concentration along certain lines of drift-flow. The masses are very striking. A number weigh 100 tons each, and one weighs between 1,000 and 1,500 tons.

The iron-ore is essentially a brown hematite, partly dehydrated at points, and from the existence of stalactitic forms it frequently bears evidence of secondary water-action. The striking thing in connection with the deposit is the occurrence of barite-crystals throughout the mass of ore. These crystals are occasionally in bunches of considerable size, but are very generally distributed.

The iron-ore as it passes farther from the source of its probable origin is improved in quality by the gradual breaking up of the boulders and the removal of the softer particles. The more distant collections of ore are higher in iron and lower in barium sulphate.

The following analyses are of samples of iron-ore from the principal "mines" which have been selected with a view to represent commercial possibilities:

	I. Per Cent.	II. Per Cent.	III. Per Cent.	IV. Per Cent.	V. Per Cent.	VI. Per Cent.
Iron, . . .	54.39	49.73	49.45	56.07	58.13	60.64
Silica, . .	1.44	3.50	3.94	3.04	2.36	2.60
Phosphorus, . .	0.075	0.05	0.064	0.086	0.086	0.125
Barium sulphate,	5.36	9.20	10.00	0.20	1.20	0.64

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Mining Operations in New York City and Vicinity.

BY H. T. HILDAGE, NEW YORK, N. Y.*

(New York Meeting, April, 1907.)

ALTHOUGH Greater New York does not bear any resemblance to a great mining district, the mining operations that are being conducted in and about the city are both extensive and interesting in character.

With regard to the extent, it may be mentioned that, at present, there is more dynamite used in New York than in any mining district in the United States; that there are under construction about 38 miles of tunnel, and that 66 miles more are projected, most of which must be constructed in the near future; that since 1902 at least 2,000,000 tons of earth has been removed in underground work, in addition to about 3,500,000 tons that has been removed in open-cut in connection with tunnel-work, and many times this quantity that has been removed in excavating foundations for buildings not connected with tunnel-work; and that there are to-day probably more than 5,000 men engaged in mining-work in and about New York City.

These undertakings include almost every operation connected with underground work, from exploration by means of trench and boring, to the sinking of large shafts and the driving of large tunnels through every kind of ground from hard rock to soft, semi-fluid silt and quicksand.

It is proposed in this paper to give a general description of some of the methods and appliances used in the opening-up of a "mine" in a large city, and a brief description of some of the "mines" now being developed in New York, connecting as far as possible the methods described with the particular work for which they were designed, in order to give a general idea of the

* Associate member of the Institution of Civil Engineers and associate member of the Institution of Mechanical Engineers.

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work that is being done and the means by which it is accomplished. Fig. 1 shows the tunnels that are now under construction. A complete and detailed description of the tunnel-work that is being done and has been done in New York City during the last few years would form an important treatise on

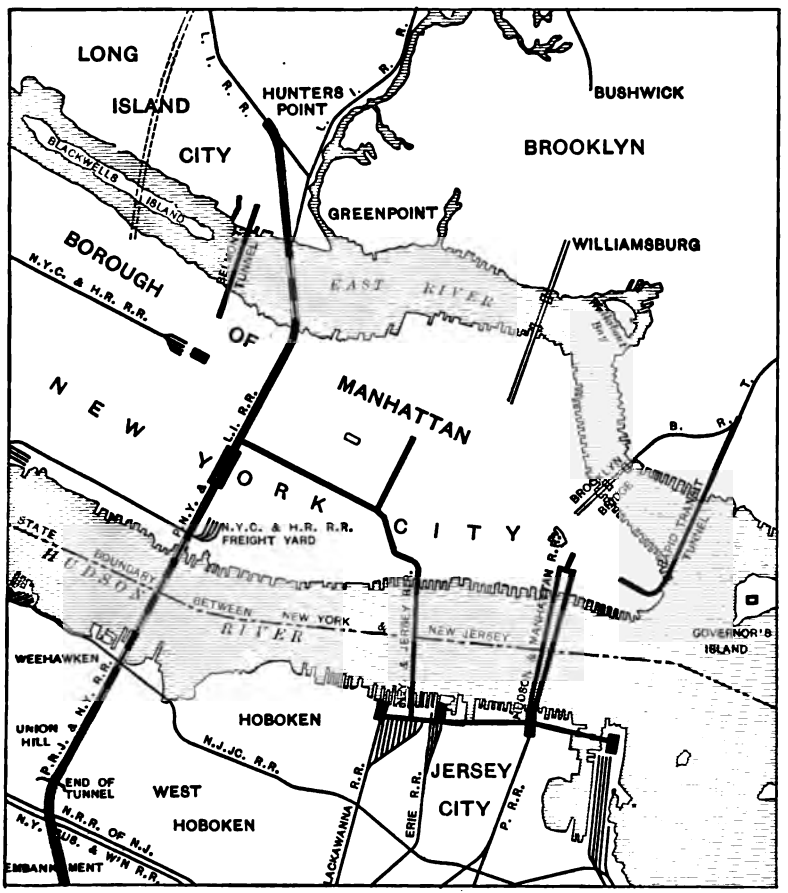

FIG. 1.—MAP SHOWING TUNNELS IN CONSTRUCTION IN NEW YORK AND VICINITY.

the art of soft-ground tunneling at least, and is entirely beyond the scope of this paper, which will, however, perhaps serve as an introduction to more complete papers recently published or to be published.

Location and Exploration.—Although the methods that are used in the exploration and development of a mine are essentially the same methods, perhaps somewhat specialized, as

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those used in the location and construction of a tunnel, and although most tunnels when completed are expected to fulfill at least one important function of a mine, that of yielding more or less profit to its owners, the considerations that govern the location and the nature of the development-work of a mine are essentially different from the considerations that govern the location, depth, size and number of tunnels.

Generally speaking, the depth, direction and position of a tunnel are, unlike the workings of a mine, fixed quite independently of the nature of the ground. The avoidance of steep grades and sharp curves, and the entrance into a terminal that is located in a position suitable to the convenience of passengers and at a convenient level, are the ruling conditions in fixing the depth, direction and position of tunnels. When the tunnel engineer becomes connected with a contemplated work, all these factors are more or less fixed, and although he may have some discretion with regard to depth, he is usually called upon to drive a tunnel in a certain position regardless of the nature of the ground, and he cannot deviate very much from the line and level given to him in order to get into more suitable strata.

The exploration-work for a tunnel differs from the exploration-work of a mine, inasmuch as its object is to explore the ground within a very limited distance of a certain imaginary line and not to seek suitable strata. This work consists of sinking a number of wash- and core-borings and trial-pits on and near the approximately fixed center-line of the contemplated tunnel, of such depth as to give as full information as possible concerning the strata through which the tunnel must pass and upon which it must lie.

It is true, as in the case of a mine, that although this exploration-work is usually carefully considered and is often quite elaborate, it very often does not reveal some very important geological feature or has to be supplemented during the course of construction.

To take one example, 147 core- and 330 wash-borings of an aggregate depth of 23,000 ft. were made to explore the ground through which the Pennsylvania tunnels were to be driven, and subsequent work has shown these borings to have been accurately taken; yet the design of the work had to be modified in some important particulars because the ground through which the borings passed was not representative of the neighborhood;

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and an additional series of borings has had to be taken from the interior of the tunnels under the North river for the purpose of exploring the ground lower down in order to discover suitable material for foundations. The design of the Weehawken shaft had to be radically changed, because in the course of excavation it was found that the contact of the trap of the Palisades and the sandstone occurred at this point. The methods of construction of the tunnels across Manhattan were modified for similar reasons. It was originally intended to tunnel from 9th avenue to 11th avenue without disturbing the surface of the ground, as the borings apparently indicated that the rock-cover was sufficient but not very heavy. It was found, however, that the rock-roof thinned out and in places disappeared, and on this account it was decided to build the tunnel by cut-and-cover methods.

When all the borings have been taken along the line of a tunnel about to be built, the center-line and grades are definitely decided upon, and, if it has not already been done, the position of the shafts is fixed and the sinking is commenced.

Shafts.—The first shaft, Fig. 2, sunk for the Hudson tunnel, in Jersey City, was of brick and about 23 ft. in diameter. It appears that the location of this shaft was chosen mainly in order to make the tunnel as short as possible. Consequently, it was built entirely in soft ground. An air-lock was built in the side of the shaft and tunneling started from there, but while the connection between the shaft and the tunnel was being completed and made safe, an accident happened that disabled this air-lock and shut off access to the tunnel. A timber caisson, Fig. 2, was then sunk alongside the shaft, and by means of this the two single-track tunnels were commenced and safely connected with the shaft, which was used for all subsequent work. The Morton street (New York) shaft for the same tunnel was a timber caisson 27 ft. 6 in. by 46 ft., and 26 ft. deep from the top of the deck to the cutting edge. This caisson, entirely in soft water-bearing ground, spanned both tunnels.

Access to the tunnels of the Pennsylvania Railroad Co. was obtained at the portal on the Hackensack meadows and through shafts located in the following places: Weehawken, in the Erie yards, near Baldwin avenue; Manhattan, between 32d and 33d streets, at 11th avenue; at 32d and 33d streets (two shafts), between Fourth and Madison avenues; between 32d and 34th

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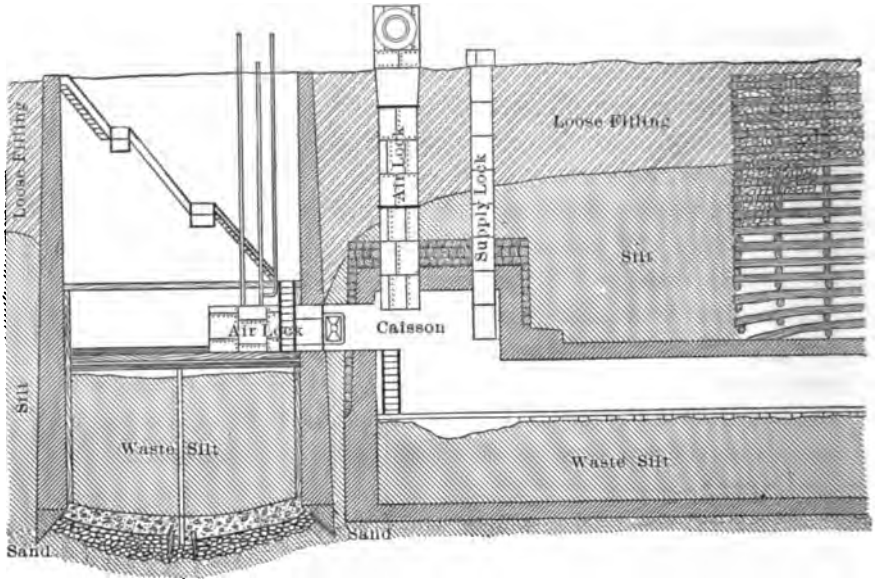

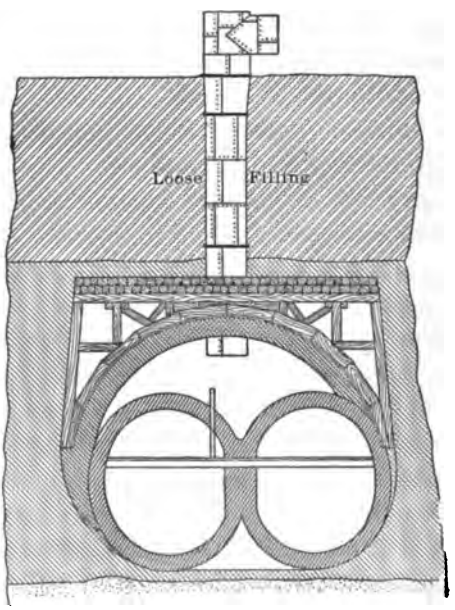

TRANSVERSE SECTION THROUGH CAISSON.

FIG. 2.—THE OLD

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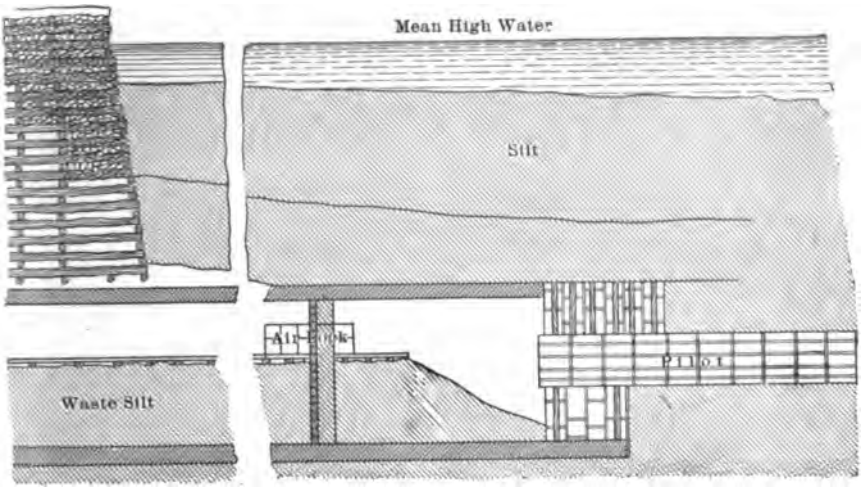
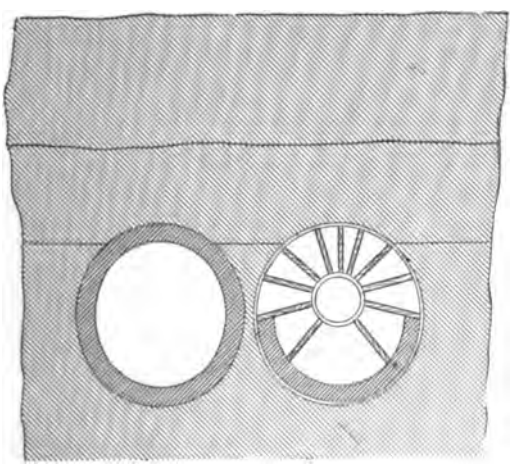


LONGITUDINAL SECTION.



TRANSVERSE SECTION.

HUDSON TUNNEL.

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streets, near First avenue; and Long Island City, one shaft in the river in Nassau slip, one on shore near the bulkhead, and others at East avenue and Borden avenue.

The considerations that fixed the positions of the river-shafts, apart from local conditions, were the desirability of shortening as much as possible the river-tunnels and of having them start out in good ground.

The Weehawken shaft of the Pennsylvania tunnel, 56 ft. by 115 ft. 9 in. at the bottom and 76 ft. deep, is lined with concrete from top to bottom. It is on the center-line of the tunnels. The ground is mostly rock. The Manhattan (North River) shaft, 22 ft. by 32 ft. and 55 ft. deep, is lined with concrete and steel down to the surface of the rock. It is about 100 ft. from the center-line between the tunnels, and is connected with them by a cross-heading. The upper part is in made ground, the lower part in rock.

The shafts between Fourth and Madison avenues, on 32d and 33d streets, 20 ft. by 30 ft. and 96 ft. deep, are not lined, being entirely in sound rock. They are 50 ft. from the tunnels they serve, and are connected therewith by cross-headings 20 ft. wide.

Each shaft for the East River tunnels serves two tunnels. The linings are 48 ft. by 74 ft., and it was originally intended if the rock were sound not to continue the linings into the rock. The shaft-linings were so designed, however, that if the rock were unsound or unreliable, the lining could be continued to the full depth of the shaft, and the upper part built up so as to be above the surface of the ground. In case it should be necessary to continue the lining to the full depth of the shaft, "eyes" were provided through which the shields could be driven, and temporarily stopped with steel plates stiffened with horizontal girders. Each shaft has double walls of steel plates about 5 ft. apart, connected and stiffened by girders, the spaces between the walls being filled with concrete. The shafts on the Manhattan side rest on the surface of the rock, but on the Long Island side it was necessary to sink them into the rock to the full depth, involving the use of compressed-air.

In general, the shafts are shallow, rarely exceeding 100 ft. in depth, and, wherever possible, arranged so that tunneling can be commenced without the necessity of using compressed-air.

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Where compressed-air has to be used in starting the heading, caissons are utilized.

Tunneling.—The difficulties of starting the heading from the shaft depend not only on the ground in which the heading will lie, but also on the ground through which the shaft has passed. Even if the heading be started in good ground, there is always the possibility that the sinking of the shaft has so disturbed the ground all the way up that the water and loose material from above may come in when an opening is made. This is what occurred at the commencement of the old Hudson tunnel. In bad ground it is always difficult to make a large vertical opening, and to protect the face at the same time. Moreover, difficulty is encountered in making a good joint between the tunnel and the shaft. In every recent case in and about New York City either the heading has been driven in good rock or the commencement has been safely accomplished under compressed-air, but when the old Hudson tunnel was commenced, several serious accidents occurred in starting the heading and in making the joint between the tunnel and the shaft.

The full-face rock-tunneling work that has been and is being done in New York City is very similar to other rock-tunneling, and it is not necessary to discuss it here further than to mention a new method that has been used for driving the heading. This method is designed for driving under streets, in the neighborhood of heavy buildings and in other places where it is necessary or desirable to reduce the shock caused by blasting to a minimum. It consists of making the first break into the solid rock face by cutting a vertical channel into it and then splitting away slabs of material by exploding vertical lines of holes on each side of the cut. This method would be very attractive, being more scientific and less brutal than firing two vertical rows of converging holes all at once to throw out the wedge-shaped piece of rock between them; but it has been found in practice that the benefits obtained even for driving headings in places where heavy blasting cannot be done, and where it is necessary to reduce shock and vibration, are not sufficiently valuable to offset the loss in time, as the same result may be obtained by increasing the number of holes and by firing relieving cuts.

The soft-ground tunneling in the neighborhood of New York

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City is unique in several ways, and a brief description of the difficulties encountered and the methods used to overcome them is of interest.

The bed of the North, or Hudson, river, composed chiefly of a compact silt containing about 33 per cent. by weight of water, flows readily under pressure. It is almost impervious to air and water and is of about the consistency of putty. This material at first presented almost insuperable difficulties to the tunnel-builders, but is now probably the easiest material through which to tunnel. The first successful method used was that of the "pilot" tunnel, illustrated in Fig. 2, which was devised for that work by Mr. J. F. Anderson, one of the original builders of the Hudson tunnel. This tunnel, the interior dimensions of which were 16 ft. wide by 18 ft. high, consisted of a very light wrought-iron shell in segments, lined with about 2 ft. of brick-work. A pilot-tube about 5 ft. 6 in. in diameter was driven from 15 to 20 ft. ahead of the working-face. The tube, made up of wrought-iron plates, was erected one plate at a time, the plate being put in and bolted up as soon as a hole had been excavated to receive it and before the silt closed in again. The thin wrought-iron shell of the large tunnel was erected in the same way, each plate being held in position by a radial strut from the pilot-tunnel. The brick-work was built inside the wrought-iron shell in lengths of 10 ft. at a time. In this way as much as 90 ft. of tunnel has been built in a month. When the work was first taken up by S. Pearson & Son, it was continued by the old methods. It became, however, more and more difficult as the deepest point in the river-bed was approached. The company therefore designed and constructed the shield to complete the work, and initiated methods that have been very successful in tunnels since built. The method of tunneling through soft ground by means of the shield-process had already been successfully used in England, and has been exclusively used in New York. It will be interesting to trace briefly its evolution as far as it has gone on the work in this vicinity.

In general, a shield for soft-ground tunneling is a means of avoiding the poling of the roof and sides of the excavation, and of facilitating and supporting the poling of the face, and forms a cover under which the lining may be erected. It con-

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sists essentially of a steel cylinder fitting rather loosely over the outside of the tunnel-lining, under the protection of which the lining can be erected and the face poled if necessary; it must be provided with some means of propulsion, usually hydraulic jacks.

In most cases an inner shorter cylinder is added for stiffness and connected to the outer by radial diaphragms. In the cells thus formed are placed the hydraulic jacks that react against the tunnel-lining and push the shield ahead. When good ground is being tunneled, the space inside the inner cylinder is left open, permitting unrestricted access to the face, except in so far as this is prevented by the vertical partitions usually necessary to stiffen the ring, and the horizontal platforms for the men to work on. If the ground is bad, the face is closed by a diaphragm in which are doors giving access to the face through the "pockets," into which the shield is divided by the vertical and horizontal partitions. When the shield is designed for ground in which it will be necessary to work in front of the shield, to pole the face, for instance, in gravel, or to remove piles, boulders or other obstructions, or to blast out rock, the outer cylinder, known as the "skin," is extended forward further at the top than at the bottom, forming a hood, under the protection of which the men can work, and which also relieves them of the necessity of caring for the roof. If the hood extends for at least two-thirds of the depth (which should always be the case), poling at the sides also is avoided and the only poling to be done is at the face.

The length of the shield depends upon the length of tunnel-lining erected at one time. In the case of cast-iron or steel lining, the "tail," or that part of the skin that can lie behind the lining, is usually made of such length that when the shield is advanced and ready for the erection of a ring, one and a half rings of tunnel-lining are under its cover. The length of the inner cylinder is determined by the length of the jack-cylinders, which is a function of the length of a ring of tunnel-lining. The length of the hood, if one is used, should be about 15 in. greater than the distance that the shield is advanced at a time. This will depend upon the nature of the ground and the length of a ring. In very bad ground, it may not be possible to advance the polings more than a few inches or a foot at a time, but in better ground a full ring-length may be made.

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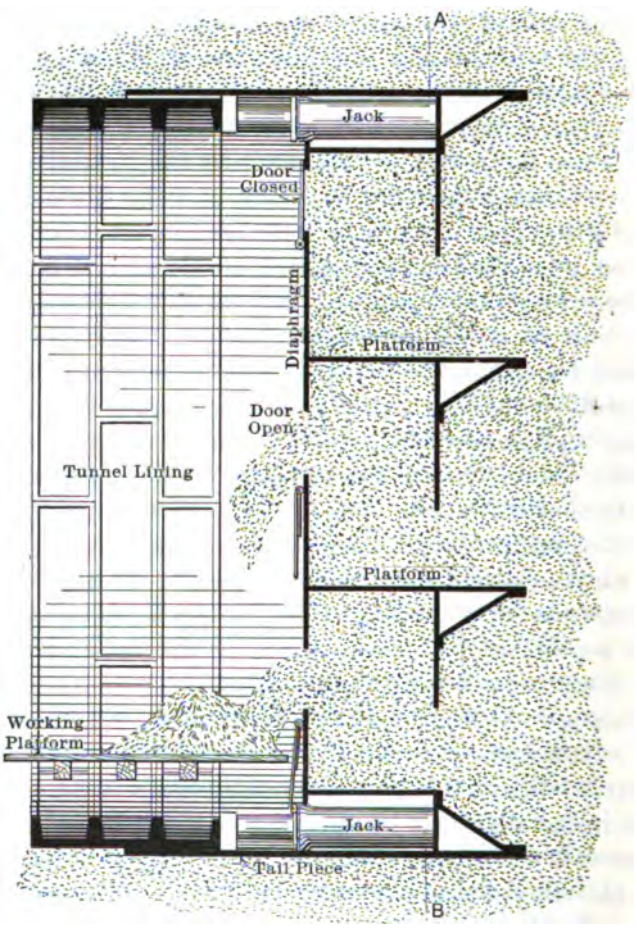
There are certain accessories added to shields, such as erectors, sliding-platforms, face-shutters, etc., that will be described with the shields on which they are used.

The shield designed by Mr. E. W. Moir and used by S. Pearson & Son in the Hudson River north tunnel, shown in Figs.



LONGITUDINAL SECTION.

FIG. 3.—SHIELD FOR CONTINUATION OF OLD HUDSON TUNNEL.

3 and 4, consisted of a steel shell, or skin, stiffened by an inner shell, and was driven by jacks. The interior was closed by a diaphragm in which were nine hinged doors. The method of working was as follows: Some of the doors of the shield were opened and power turned on to the jacks. The shield moved

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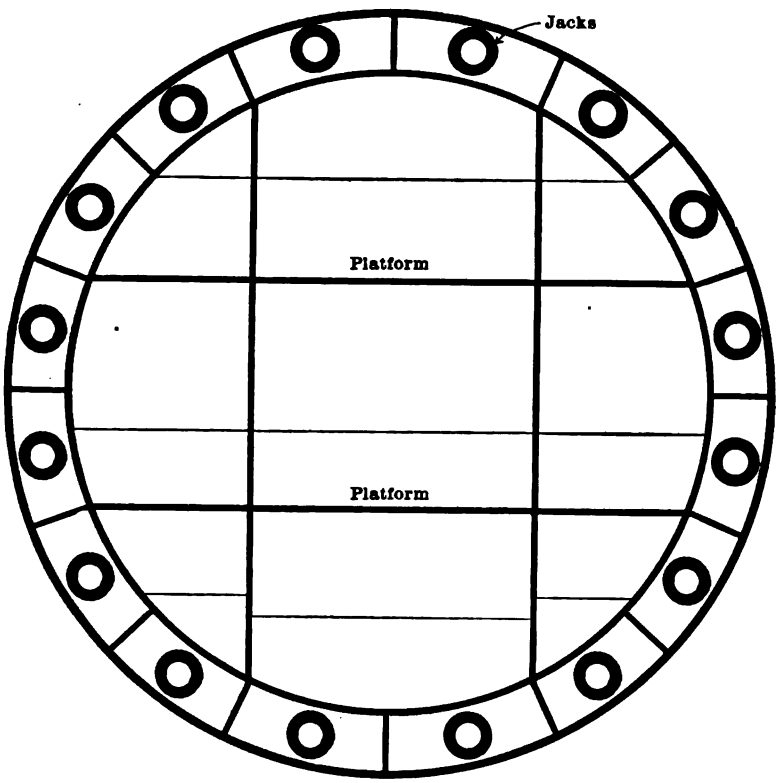
slowly ahead, laying bare as it went the " tail," which had been lying over the outside of the lining, and ultimately giving, when the jacks were returned, space for the erection of another ring. The silt squeezed slowly in through the open doors and falling upon the platform was loaded into cars and removed. As it moved slowly ahead, the progress of the shield was gauged at various points around the circumference, and when



SECTION A-B.

FIG. 4.—SHIELD FOR CONTINUATION OF OLD HUDSON TUNNEL.

it had traveled the full length of a ring, the doors were closed, the invert cleaned out and the ring erected. This was done by means of a hydraulic erector that trailed behind the shield on rails supported by the iron lining. This erector consisted of an arm revolved on a pivot by two hydraulic jacks, and carrying a third hydraulic jack, the ram of which was fitted with a jaw capable of being attached to one of the tunnel-segments.

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The whole arrangement was mounted on a carriage with four wheels.

The shield method of tunneling proved much more successful than the pilot-tunnel method, and with it from 7 to 10 ft. per day were built in the Hudson north tunnel.

A modification in the method of working the shield was made when it was found possible later in building the south tube (Hudson tunnel) to push the shield ahead with the doors closed, consequently taking in very little if any silt. This modification reduced the operation of tunneling to two parts: shoving the shield ahead and erecting the iron ring. As much as 50 ft. of tunnel per day was built in this manner. Later, in the Pennsylvania tunnels it was found impossible to control the shield and maintain the line and the grade if the shield were shoved with closed doors, and for good work it was necessary to take in a part of the silt through the doors. If this were not done, the shield rose above the grade and could not be kept down, and the tunnel itself rose considerably after construction. The amount taken in varied from 20 to 100 per cent. of the volume of the tunnel. The North River silt is so mobile that it would be quite easy to raise or lower considerably a tunnel built in it, and thus change or correct grade and perhaps alignment.

With the methods above described the process of tunneling through a full face of silt has become safe, quick and easy, and it does not appear that much further improvement can be looked for except in mechanical details—handling muck, shield, rams, and erecting iron—and organization.

When the shield was not entirely in silt, difficulties appeared that necessitated further revision of methods and considerably reduced the rate of tunnel-building. When the north Hudson tunnel had been driven about 4,000 ft. it was found that there was a ridge of rock through which the tunnel would have to be cut, and it turned out subsequently that the rock rose in places as much as 16 ft. above the invert. This was the first time in the history of tunnel-driving that it had been necessary to blast out part of a face of rock, and at the same time prevent an inflow of soft earth. To overcome this difficulty a horizontal platform was built in the shield about 8 ft. above the invert and projecting about 2 ft. in front of the shield, sup-

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ported by raking struts from below, to carry the roof of the excavation. Then that portion of the silt between the platform and the surface of the rock was held up by poling-boards strutted to the shield, while the rock in the invert was blasted out with light charges of dynamite.

This method of working was generally satisfactory, and by its use from 2 to 4 ft. of tunnel were built per day. One difficulty that frequently occurred in connection with this work was what is technically called a "blow." The air-pressure in the tunnel, about 35 lb. per sq. in., was found to be satisfactory when ordinary progress was made, any lowering of this pressure allowing the silt to flow excessively. If, for any cause, the shield was kept standing in the same place for a day or two, the silt seemed to soften considerably for some reason or other, with the consequence that when progress was resumed and the shield shoved ahead, the air escaped, usually between the tunnel-lining and the shield-skin, with a loud roar, carrying with it the soft silt, and opening up communication with the river; the pressure in the tunnel fell, the atmosphere became foggy, and unless the escape of air was speedily stopped by stuffing bags of sawdust, clay, or other material into the hole, a point was soon reached when the escape of air began to be accompanied by a heavy inrush of water. This was followed by a rise in the pressure which soon caused another blow, and so on until the tunnel was full of water.

These occurrences were rather alarming and delayed the work considerably, but very seldom, if ever, caused loss of life. They could be and were either prevented, if they seemed imminent, or cured, after the tunnel was flooded, by blanketing the river-bed at the spot where the "blow" occurred. One or two barge-loads of good clay were dumped just over the shield, and this replenished the "cover" that had been or was in danger of being blown away.

Another trouble, and one that was doubtless partly responsible for the occurrence of blows in the Hudson tunnels, was that every time the shield was advanced the face was lost and the silt flowed into the "pockets" of the shield and through the doors. This delayed the work by making it necessary to clean out the pockets of the shield every "shove," and contributed to the production of blows by denuding the tunnel of

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just that amount of "covering" clay. This trouble was avoided in the Pennsylvania tunnels by the method of working there adopted. The face above the rock was held by vertical poling-boards, and as these were advanced they were strutted against the shield. When the face was completely poled, horizontal walings were put against the polings and held in place by the sliding-platforms provided, and by struts which passed through the shield against the tunnel-lining. The necessity for supporting the poling independently of the shield was recognized early in the driving of the Blackwall tunnel in London, England, and it has been used in every tunnel of that kind driven in England since. The design and position of the shield (the shield had rolled on its axis about 45°) in the Hudson north tunnel prevented it being used there, but it was used in the Morton street extension and probably in the Hudson south tunnel.

The methods of working described above refer only to tunnels built entirely in silt or part in rock and part in silt, but entirely in ground that is impervious to air and water.

Some portions of the North River tunnels of the Pennsylvania Railroad Co., the Morton street extension of the Hudson tunnel, the Pennsylvania Railroad Co.'s East River tunnels and the tunnels from the Battery to Brooklyn are being built in sand and gravel of various kinds, which permit the escape of air and the inflow of water, and must consequently be dealt with in a special manner.

As has already been stated, with a full face of silt, no excavation is done. The shield is simply pushed ahead, and the silt removed is that which flows in through one or several doors. No work is done ahead of the shield.

With a full face of sand or gravel, almost all the material has to be excavated, and very little of it can be displaced, necessitating the maintenance of a vertical face ahead of the shield, either by poling-boards or by other means.

When the face is part silt and part rock, the air-pressure in the tunnel is generally lower than the pressure of the silt, and the poling-boards, etc., are depended upon to exclude the silt. If there is a small leak of silt it does not matter.

When the face is part sand or gravel and part rock, the object aimed at is to make the air-pressure balance the water-pressure; and since the water-pressure is greater at a lower

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point of the face than at a higher, it follows that over part of the face the air-pressure is generally too great, and over the remainder too low. The difficulty is to prevent the excessive escape of air on the one hand, and the excessive inflow of ground on the other. In the case of very fine sands, such as occur extensively in the bed of the East river, excessive escape of air is accompanied by the removal of the material; the falling outwards or shrinking of the face, and ultimately by the opening up of a passage to the river, and the occurrence of a "blow." In the case of coarse sands and gravels, this latter difficulty (the removal of material) does not occur, but in this case the ground is so open that the escape of air is always excessive unless a very small portion of the face is exposed at one time. If (in fine sand) the air-pressure were much lower than the water-pressure, it would be impossible to prevent the inflow of the fine sand, which, when saturated with water, is more mobile than the silt, and the polings would collapse.

It is evident that the adjustment of the air-pressure in a tunnel passing through sand or gravel is much more delicate and difficult than in a tunnel passing through silt.

With regard to methods in the Morton street extension, and in those parts of the Pennsylvania tunnels that are in sand and gravel, the method of working was similar to that described for a face part silt and part rock, except that horizontal polings were used instead of vertical polings. The top board was advanced first and strutted to the shield, and then the second, and so on until the whole face was advanced, when "soldiers" were put against it and held by the sliding-platforms and by struts that passed through the shield to the tunnel-lining.

In the East River tunnels of the Pennsylvania Railroad Co. steel shutters were provided to be used instead of poling-boards. These shutters consisted of steel plates stiffened with angles and provided with small doors, running in guides, which were attached to the compartment-walls of the shield. They were controlled by long screws fixed to their ends and running through bearings attached to the compartment-walls. The top shutter was intended to be advanced first, the material being raked out through small doors. Then the second one could be advanced, the material being removed through the space between the top one and the second one, and so on. This is the

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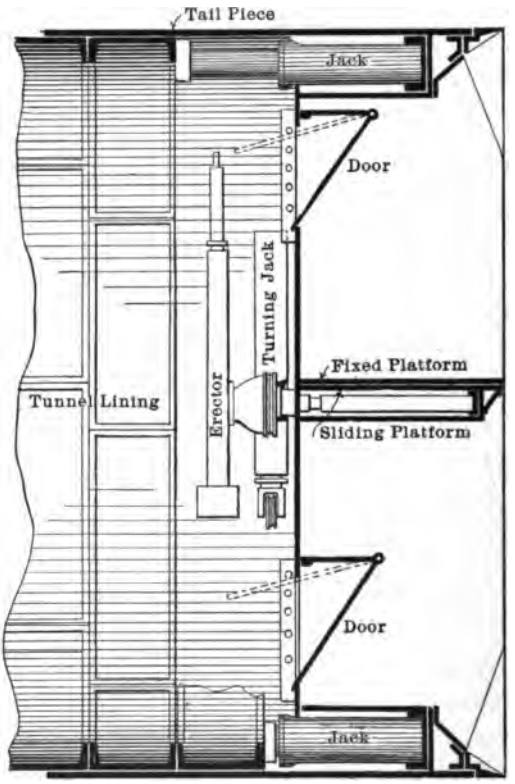
way in which a similar shield was used for the Blackwall tunnel, with excellent results.

In the East River work, the shutters have not, so far, been used in this way, but the face is poled with horizontal poling-boards down to the level of the lower platform, the lower part of the shield being left practically full of sand. These polings are then held up with struts that will telescope under considerable pressure while the shield is advanced.



LONGITUDINAL SECTION.

FIG. 5.—SHIELD FOR HUDSON RIVER SOUTH TUNNEL.

This is, in effect, the same method that was used in the North River tunnels.

To sum up, it is evident that the method used of working a sand- or gravel-face consists of poling the face with horizontal polings, and then holding it independently while the shield is advanced. An attempt was made to use vertical polings in the North River tunnels, but was abandoned. The method thus

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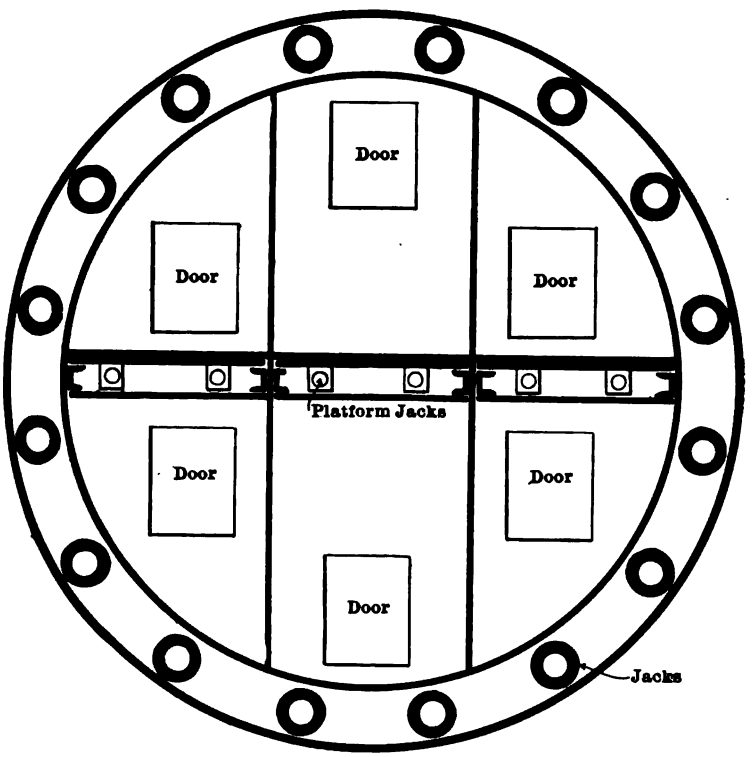
used was probably originally used in part of the Blackwall tunnel work, and with modifications was used in coarse gravel in the Baker street and Waterloo tunnels. It appears to be the best, if not the only satisfactory method of working with a face of sand or gravel.

At this point a brief description of the various types of shields that are being used may be of interest.

The shield for the north tunnel of the New York & Jersey



TRANSVERSE SECTION.

FIG. 6.—SHIELD FOR HUDSON RIVER SOUTH TUNNEL.

Railroad Co. has been mentioned. It consisted of a steel shell or skin, 19 ft. 11 in. in exterior diameter, 10 ft. 6 in. long and 1.25 in. thick, stiffened by the addition of an inner shell, 3 ft. 2 in. long and 16 ft. 11 in. in internal diameter, connected to it by radial webs. The compartments of this ring were utilized to hold the 16 hydraulic jacks used to propel the shield, utilizing the iron lining as an abutment. The interior, shut off

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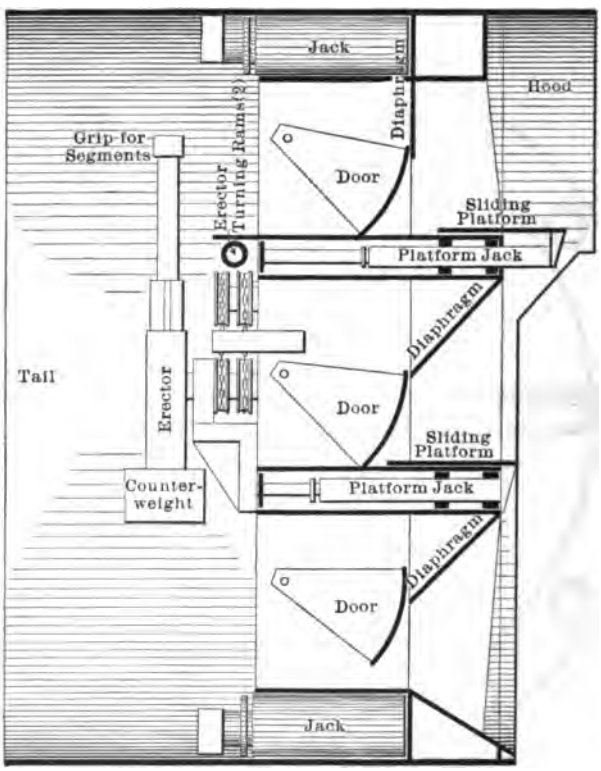
from the tunnel by a diaphragm, was divided into nine pockets by two vertical partitions and two horizontal platforms, and these pockets were open to the face and accessible from the tunnel through hinged doors.

The south tunnel shield, Figs. 5 and 6, consisted of an outer skin, 16 ft. 9 in. in internal diameter and 10 ft. long, with an inner shell of 14 ft. 9 in. internal diameter and 3 ft. long. The



LONGITUDINAL SECTION.

FIG. 7.—SHIELD FOR NORTH RIVER TUNNELS, PENNSYLVANIA RAILROAD.

space inside this shell was divided into six compartments, or pockets, by two vertical stiffening partitions and one horizontal platform. The face was closed with a diaphragm, which was provided with six doors that swung inwards on a pivot. A horizontal sliding-platform in three parts was provided, each part being actuated by two hydraulic jacks.

The erector used for placing the lining in position, which was on the shield, consisted of a long arm carrying a hydraulic

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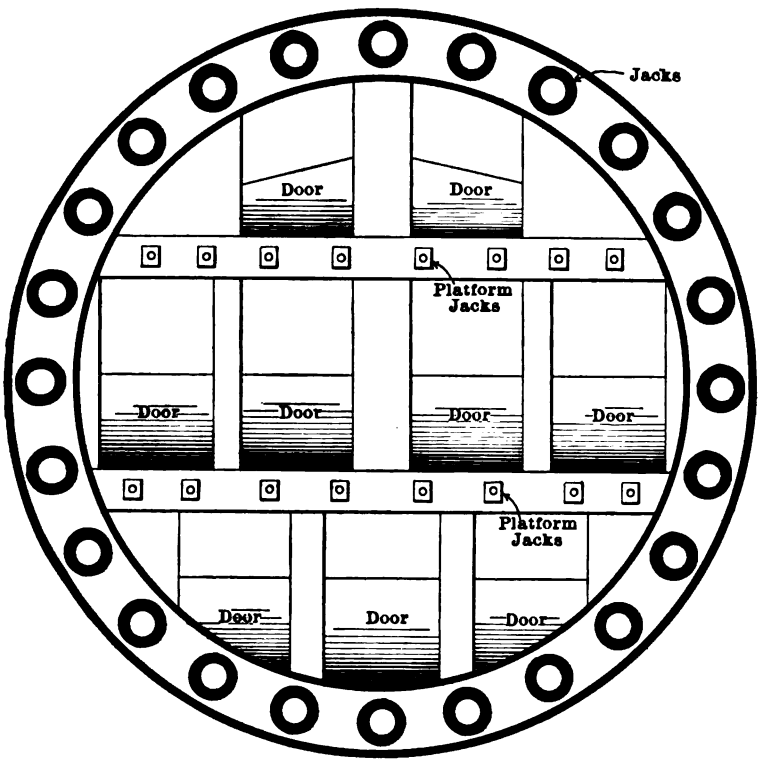
jack, which was caused to rotate about an axis coincident with the axis of the shield by two other hydraulic cylinders. The shield was driven by 16 hydraulic jacks, 8 in. in diameter, suitable for use with a pressure of 5,000 lb. per sq. in., giving a maximum pressure of 125 tons per jack, or 2,000 tons for the whole shield.

The shields that were used for the North River tunnels of the



TRANSVERSE SECTION.

FIG. 8.—SHIELD FOR NORTH RIVER TUNNELS, PENNSYLVANIA RAILROAD.

Pennsylvania Co., Figs. 7 and 8, had an external diameter of 23 ft. 6¼ in.; the skin being made up of one ⅜- and two 0.75-in. plates, the internal diameter was 23 ft. 2 in., allowing a clearance of 1 in. all around the 23 ft. tunnel-lining. The total length of the shield was 15 ft. 11.5 in., and the hood, which covered rather more than one-third of the circumference, extended 2 ft. 3⅜ in. further. The length of the tail was 6 ft. 4.5 in., being long

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enough to cover two and a half rings of the lining 30 in. long. The forward part of the shield was divided into nine pockets by two vertical partitions and two horizontal platforms. Each of the platforms had a sliding extension made in four pieces, each piece being actuated by two hydraulic jacks.

The pockets were closed at the end nearest the face by plates covering about half the opening and inclined so that the upper edge of the plate was nearest the face. The remainder of the opening could be closed by means of a door. The doors were segments of a cylinder, and swung on pivots attached to the vertical diaphragms. A single erector was used, which, being typical of the erectors used on the shields described, is shown in Fig. 9. The shield was driven by 24 jacks, with rams 8.5 in. in diameter. The maximum hydraulic pressure available was 5,000 lb. per sq. in., and this gave, with all jacks, a total forward pressure of 3,300 tons.

The East River (Pennsylvania tunnels) shield, of which the leading features are shown in Figs. 10, 11, 12 and 13, is of very similar design to the one used for the Blackwall tunnel. It is 18 ft. long over all, and is provided with a hood that can be advanced and withdrawn at will. The face is closed by two diaphragms fitted with air-locks, the object being to permit of a higher pressure of air being used in the face than in the tunnel.

The shield is fitted with two erectors, and the erection of the iron lining can go on at both sides of the tunnel at the same time. The erector-arm in this case is revolved by a single cylinder, the ram being fitted with a rack which gears with the toothed hub by which the revolving arm is mounted on its pivot. In the erectors on other shields mentioned there were two cylinders for turning the arm, and the transmission was either by cable or by chain. The face-shutters with which this shield is fitted have already been briefly described. There are 27 jacks, with 9-in. diameter plungers, for shoving the shield, and they are capable of exerting a total forward pressure of 4,050 tons with 5,000 lb. per sq. in. pressure. They are more closely spaced in the bottom than in the top. The reason for this arrangement has already been suggested: if the air-pressure were so regulated as to balance the water-pressure at the

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bottom of the shield, it would be about 10 lb. per sq. in. in excess at the top, and "blows" would result; if it be less than sufficient, as is the case, the invert of the shield and tunnel is filled to a corresponding level with sand and water, and below this level the face cannot be poled. Additional power is therefore necessary to displace this material in the bottom when the shield is advanced.

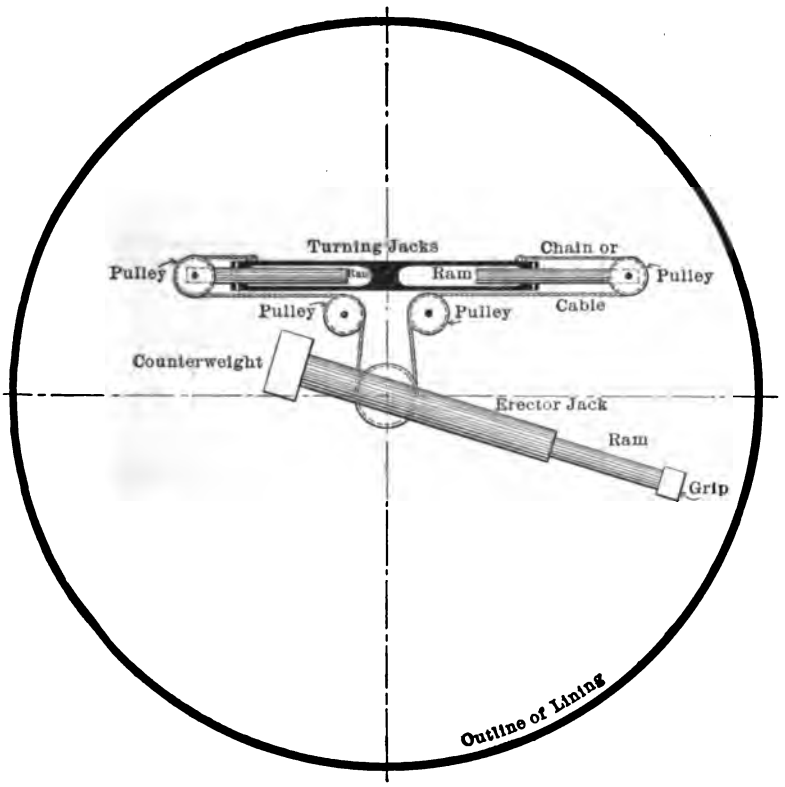

FIG. 9.—ERECTOR FOR IRON LINING.

A cross-section of the iron lining used on the Pennsylvania tunnels is shown in Fig. 14 as a type.

The shields, Figs. 15 and 16, that have been used in the Battery tunnel of the Rapid Transit Commission differ from others in several respects, the most important of which is, probably, the lack of provision for vertical protection of the face. The shield was provided with a fixed horizontal platform, and arrangements

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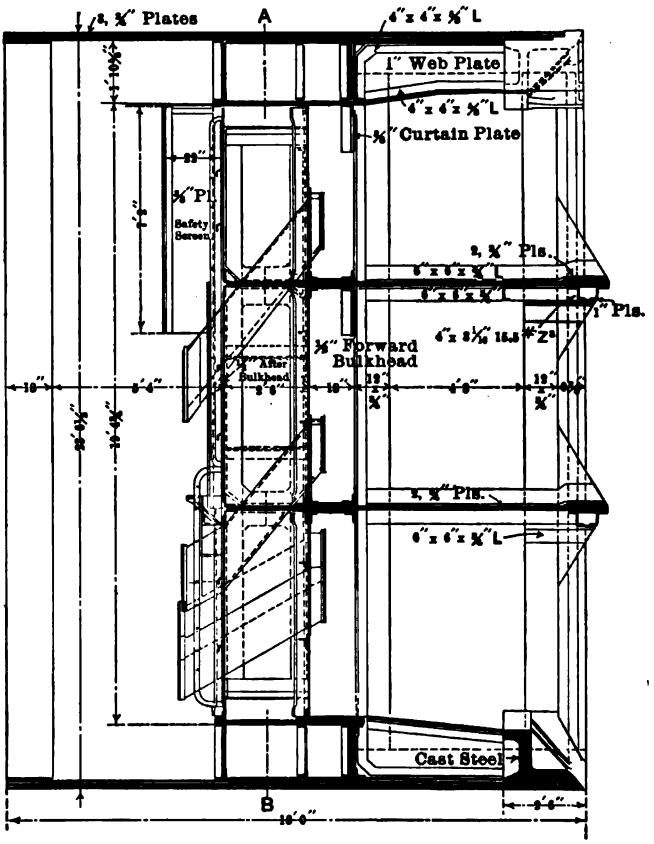
were made for other horizontal platforms, to be made of poling-boards, and on these the material was allowed to fall with its natural slope. The erector was mounted on a separate carriage that trailed along behind the shield. It was rotated by a small three-cylinder compressed-air motor, and the hydraulic cylinder in the arm received power, as did the shield-jacks, from a



LONGITUDINAL SECTION.

FIG. 10.—SHIELD FOR EAST RIVER TUNNELS, PENNSYLVANIA RAILROAD.

compressed-air-driven power-pump mounted on the same carriage.

The cost and the rate of driving tunnels through loose ground by the shield-method vary very considerably, and are dependent upon a number of considerations. Of course, the most important is the nature of the ground. In a soft clay or silt, where it is sometimes possible to shove the shield ahead without re-

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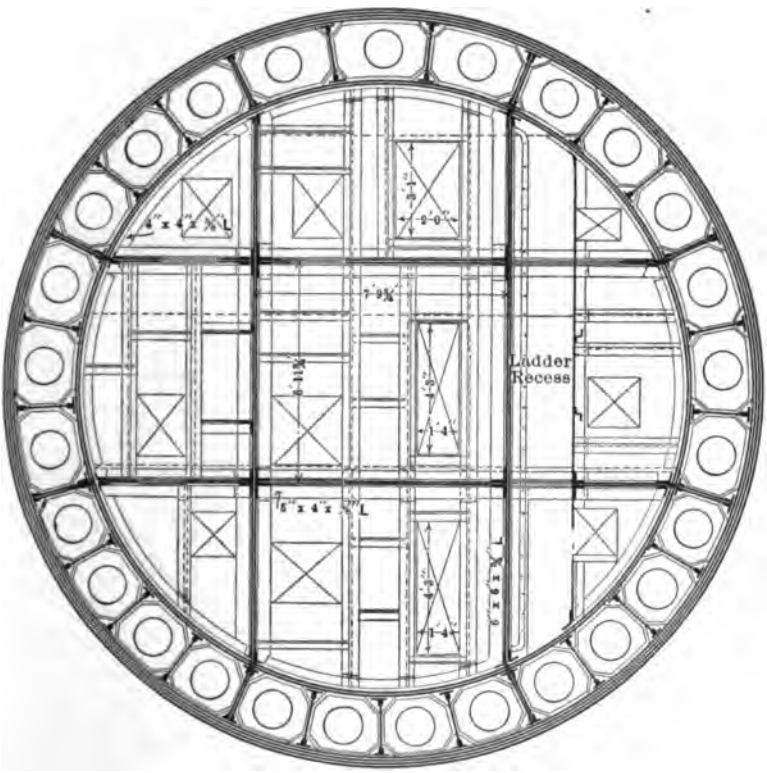
moving through the tunnel any material whatever, the labor-charges are very low.

In a sand or gravel fully charged with water where all the material has to be removed, and the face kept by polings, and especially where part of the face is rock, the cost is much higher. A higher air-pressure has to be maintained, with consequent increase in the price of labor. The labor is not so efficient, the



SECTION A-B.

FIG. 11.—SHIELD FOR EAST RIVER TUNNELS, PENNSYLVANIA RAILROAD.

progress is very much less, and often it is quite irregular. For instance, in the Hudson tunnel at one time there were two faces at which work was proceeding, one of sand and the other of silt. The shields were alike and the tunnels the same size. The progress in the first was at one time not much more than 1 ft. per day, although this was afterwards increased to 6 or 7 ft. per day. In the second, the progress was at the rate of 50 ft. in 24 hours.

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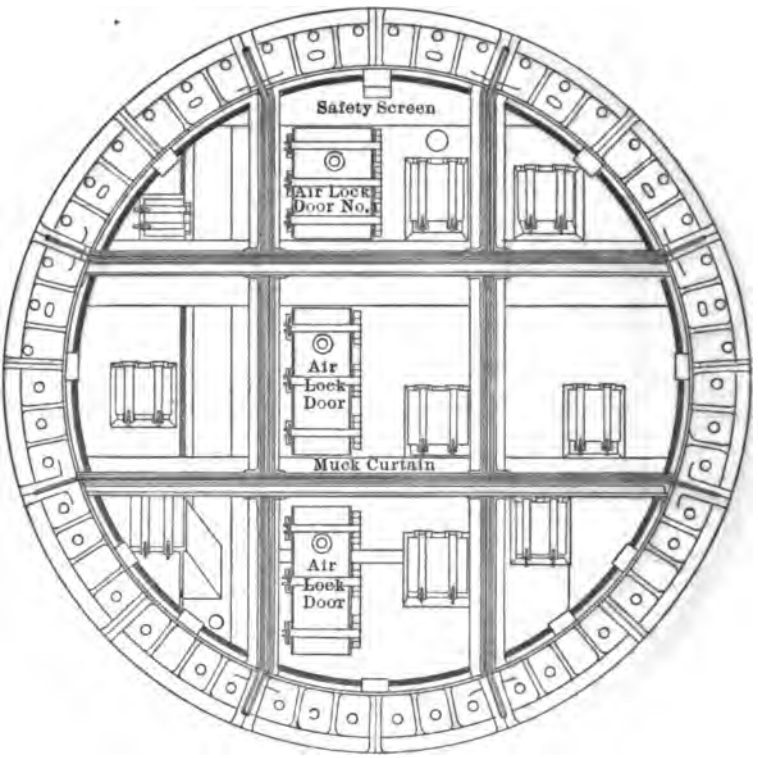
The cost of labor per day was probably about the same in each case, so that the labor-cost per unit of length was evidently very different in the two cases.

The design of the shield is another matter that has very great influence on the progress made. Of necessity, every shield that has been used up to the present time has been more or less experimental, and many of them have had to be changed



FRONT VIEW.

FIG. 12.—SHIELD FOR EAST RIVER TUNNELS, PENNSYLVANIA RAILROAD.

or added to in some respect or other during the course of the work. There seems to be no reason, however, with the lessons that have been learned under the North and East rivers, together with the experience that has been obtained before, why a shield should not be designed that would be suitable for any conditions that might arise during the progress of the work.

Such a shield would have a hood extending down over about two-thirds of the circumference, and forward about 4 ft. from the cutting-edge. It would have strong and efficient sliding-

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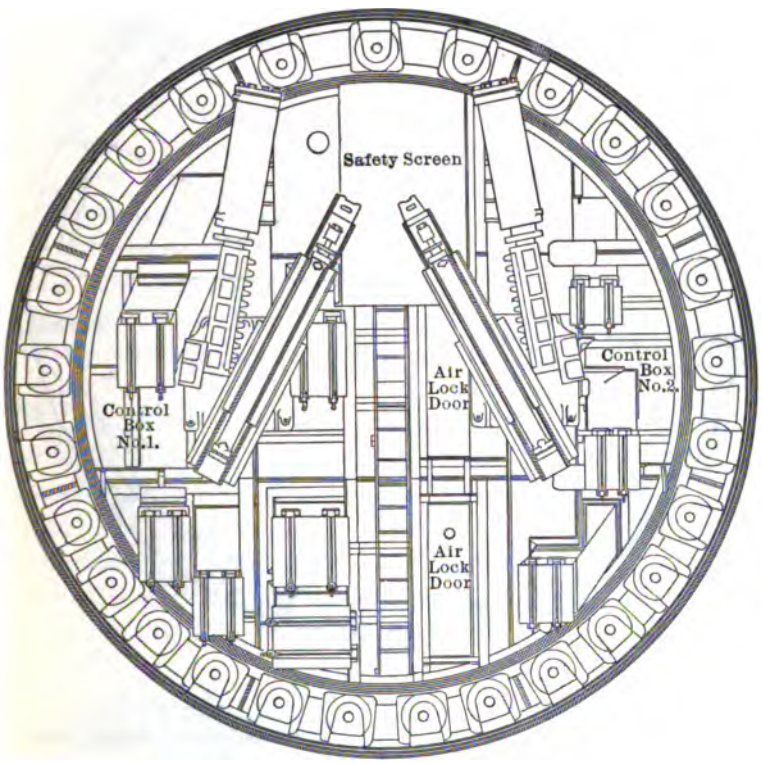
platforms. It would be provided with some means of supporting the face, so that the shield could be advanced without any change in the arrangement of the supporting-pressure. The door-space would be as large as could be obtained, and the openings would be fitted with doors that could be kept open or closed, as was required by the method of working and the nature of the material, and that were capable of being surely and



REAR VIEW.

FIG. 13.—SHIELD FOR EAST RIVER TUNNELS, PENNSYLVANIA RAILROAD.

quickly closed in an emergency. All the openings would be fitted with a removable siphon-trap arrangement for the protection of the men and tunnel in the event of a blow in the face. As regards the propelling-mechanism, the jacks should be provided with a good automatic "pull-back" that would avert the troublesome necessity of prying and wedging back the rams in order to erect the iron.

The labor question is one that has probably a greater effect

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on the cost of tunneling than is the case in other kinds of engineering work. The progress of a tunnel is peculiarly liable to interference caused by unskillful or unsympathetic work in the face. It has been necessary in every tunnel that has been

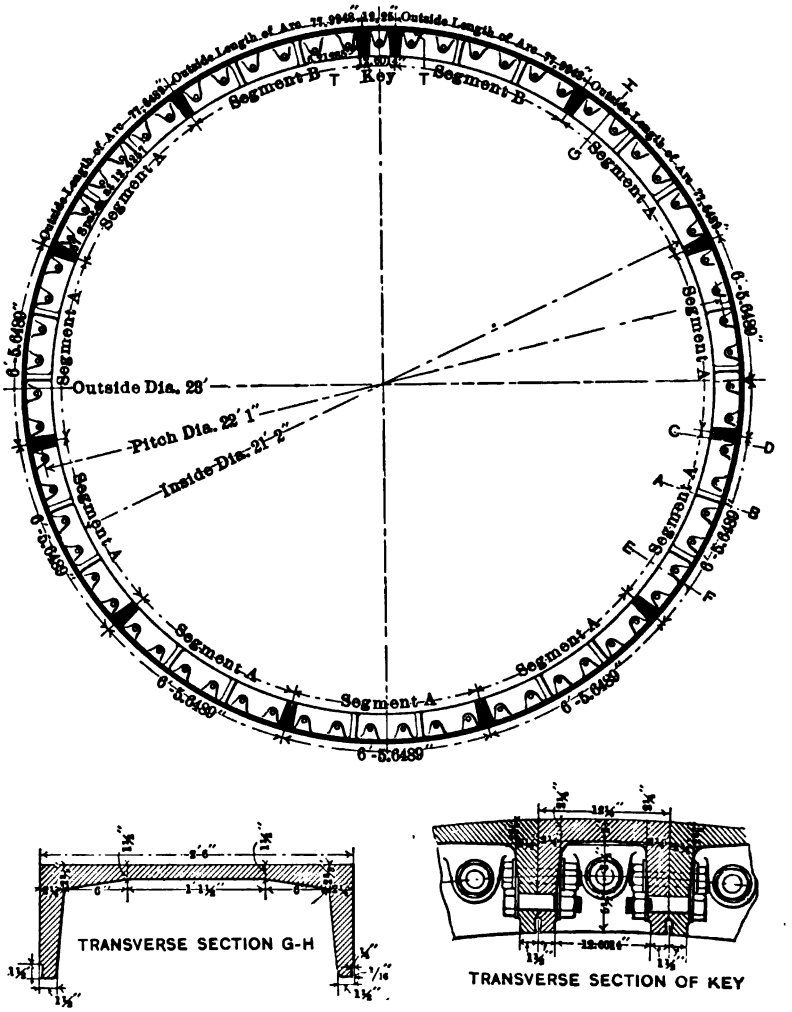

FIG. 14.—TYPE-SECTION OF IRON LINING.

built in New York to train the men in their work, and to produce from the rawest material more or less skilled tunnel-builders. This has increased considerably the cost of supervision, and has added materially to the natural difficulties of the work. The task has not been at all simplified by the ten-

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dency to "unionize" the laborers and foremen, nor by the tendency of the sensational newspapers to magnify the "heroism" of the men.

The depth of the tunnel below the surface, in addition to increasing the cost of raising and lowering material, usually affects the air-pressure it is necessary to use. An increase in the air-pressure is accompanied by an increase in the daily rate of pay and by a diminution of the number of hours worked, and, since the work proceeds night and day, by an increase in the number of men employed.

The size of the tunnel has its effect on the efficiency of the work, but it is not possible here to make a full discussion of this point. It may perhaps be proper, however, to emphasize the somewhat obvious fact that the work will be more efficient in a tunnel where it is possible to work a great number of men in the face, conveniently, at one time.

The following figures, showing the approximate rates of progress attained in the various tunnels named, may be of interest:

Hudson River North Tunnel (Old Hudson Tunnel): a brick tunnel in silt, 16 ft. wide by 17 ft. high internally, driven by pilot-tunnel method at the rate of about 3 ft. per day. Iron-lined shield-driven continuation of above tunnel, 19 ft. 6 in. external diameter, in silt, at the rate (exclusive of contingencies) of from 10 to 12 ft. per day; in part rock and part silt, from 1 ft. 8 in. to 3 ft. 4 in. per day. From 18 to 20 men at work at the shield.

Hudson River South Tunnel (16 ft. 7 in. external diameter): an iron-lined shield-driven tunnel constructed through silt at the rate (exclusive of contingencies and delays for the lack of iron lining) of about 50 ft. per day.

Morton street extension of Hudson Tunnel: 16 ft. 7 in. external diameter, in sand, driven at the rate of from 1 to 8 ft. per day. From 18 to 20 men at the shield.

Pennsylvania North River Tunnels: 23 ft. external diameter, in rock, or face of part rock and part silt, from 2 to 3 ft. per day average; full face of silt under river, from 13 to 14 ft. per day average. About 30 men at the shield.

The Pennsylvania East River tunnels are still being driven. They are 23 ft. external diameter, and the progress has varied

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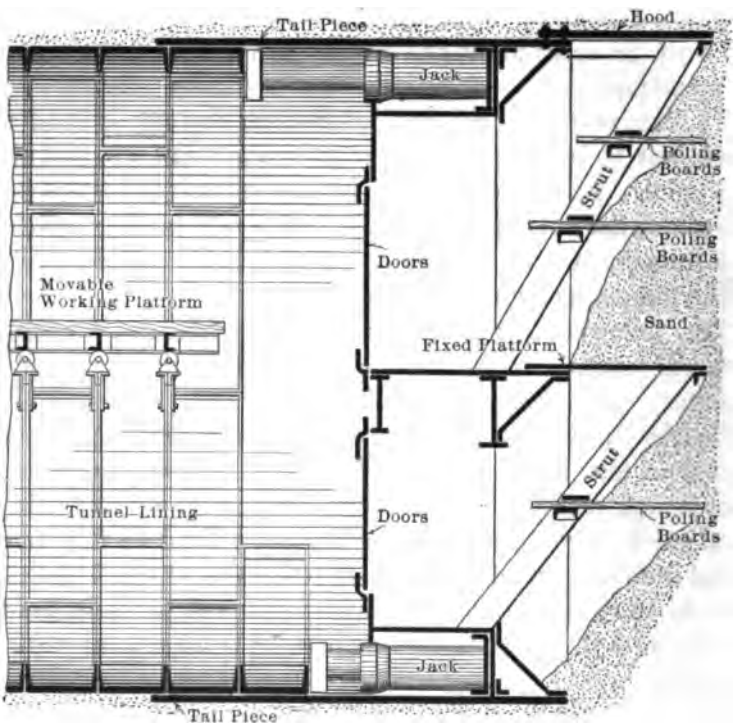
from a few inches to 12 ft. per day, according to the nature of the ground passed through.

Battery Tunnel of Rapid Transit Commission: 16.7 ft. external diameter, in sand, made maximum progress of 12 ft. per day. About 10 men at the shield.

The rate of pay for tunnel-laborers in compressed-air is from $3 to $4 per day of from 8 hr. in pressures below 32 lb. to



LONGITUDINAL SECTION.

FIG. 15.—SHIELD FOR BATTERY TUNNEL.

6 hr. in higher pressures. These figures are, of course, approximate, being dependent upon varying conditions encountered.

The work here described is distinguished from ordinary mining work by the fact that it is carried on with the aid of compressed-air, which assists to prevent the inflow of the fluid or semi-fluid earth. A bulkhead is built in the tunnel or shaft, separating the face of the work from the outside air and forming a chamber into which air is pumped at the necessary pres-

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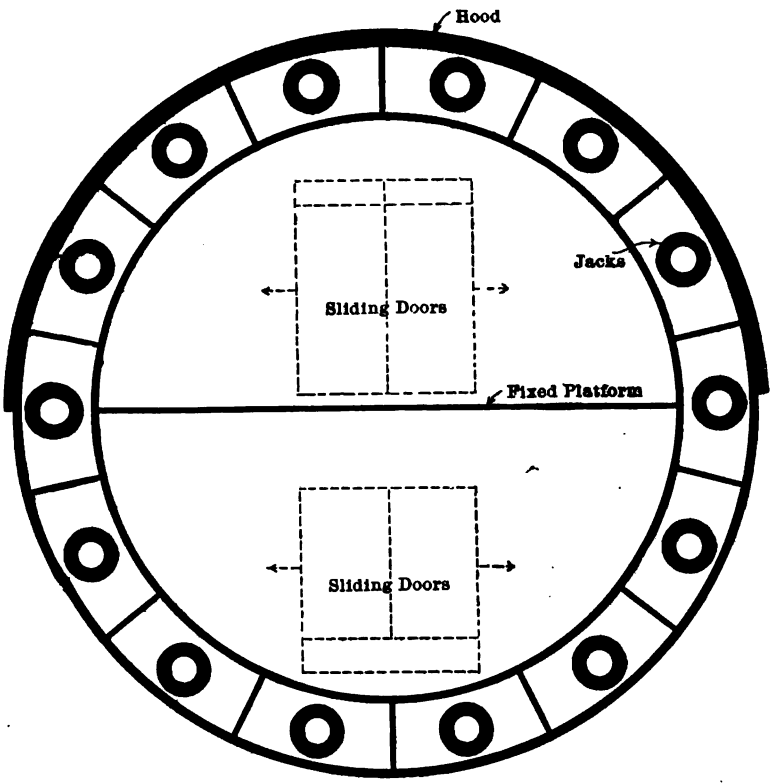
sure. Access to this chamber is obtained through air-locks, which are usually steel cylinders fitted with air-tight doors opening towards the face and with pipes and valves, by means of which the pressure inside the locks can be varied at will from normal pressure to the pressure inside the chamber.

The pressure inside the chamber is dependent upon the depth of the tunnel below the surface of the ground, being in



TRANSVERSE SECTION.

FIG. 16.—SHIELD FOR BATTERY TUNNEL.

the case of silt equal to or less than the head of silt and water above the tunnel, and in the case of sand or gravel equal to the head of water a few feet above the invert of the tunnel.

The quantity of air necessary, and consequently the capacity of the compressor-plant, depends, in the case of a tunnel in silt or other air-tight material, upon the number of men working in the tunnel, the object being to give efficient ventilation everywhere, and to maintain the air of such purity that analy-

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ses do not show the presence of more than 0.1 per cent. of carbon dioxide. This may take as much as 100 cu. ft. of free air per man per minute. In the case of sand, gravel, or other open ground, sufficient air is required to replace that which escapes through the face, and the amount depends upon the closeness of the ground and the manner in which the face is worked.

The effect of compressed-air on the health of the men is a matter for doctors rather than engineers; experience has shown that work can be carried out without risk to life or injury to health in pressures up to 35 lb. per sq. in., provided certain precautions are taken. These are: (*a*), the exclusion of men not in good physical condition, especially when suffering from heart or lung troubles or catarrhal affection of the head; (*b*), efficient ventilation of the tunnel, particularly where men are at work; (*c*), provision of warm clothing and warm quarters for the men on leaving the tunnel, and, in general, the avoidance of chills; (*d*), the careful observance by the men of all the ordinary rules of health.

There is one method of tunneling that has been tried experimentally, and is worthy of mention—viz., the "freezing" process. This process, which has been successfully used many times in shaft-sinking, has never been applied to driving a long tunnel. It was used successfully in passing through a short length of bad ground in the Stockholm tunnel, and is now being tried in a tunnel in Paris. The problem of tunneling by freezing is essentially different from shaft-sinking by freezing. The face is vertical instead of horizontal, and the length is great and indefinite. In the case of a shaft the freezing-pipes can be sunk outside the excavation and all efforts concentrated in freezing an impervious wall about the ground to be excavated. In the case of a tunnel, it is almost unavoidable that the freezing should take place from the center, and in consequence the ground that is frozen the hardest and is strongest is the ground that has to be excavated, and would perhaps have been better if soft. The weakest part of the frozen mass is the wall outside the excavation—the part where strength is most necessary.

Among the contractors' proposals to construct the East River tunnels of the Pennsylvania Railroad Co. was one to drive them by the freezing-process, and the method suggested

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FIG. 17.—VIEW OF BRINE-PIPES IN PILOT-TUNNEL.

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FIG. 18.—VIEW OF PILOT-TUNNEL DURING REFRIGERATION.

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was first to drive a small pilot-tunnel by any ordinary method, and then from that to freeze the ground out to such a distance that when the larger tunnel was excavated a sufficient thickness of frozen material would remain to resist the pressure of the water and the unfrozen ground beyond. The Pennsylvania Railroad Co. thereupon, with characteristic enterprise and forethought, made an experiment in tunneling by the freezing-process at considerable cost.

A shaft was sunk to a depth of about 85 ft. through the East 35th street pier, and a tunnel 7 ft. 6 in. in diameter was driven out a distance of 165 ft. by means of a small shield and compressed-air. A number of pipes were attached to the lining with their axes parallel to the axis of the tunnel, and brine at a temperature of about 35° below zero, Fahrenheit, was circulated through them for several months. Careful observations were made of the temperature of the ground surrounding the pilot-tunnel and of the quantities of heat extracted by the brine, and laboratory determinations on a large scale were made of the thermal conductivities of different kinds of sand and gravel. The object of the experiment was not merely to test the feasibility of constructing a large tunnel by the freezing-process, but to collect, in addition, such information as would enable reliable estimates to be made of the time and the cost of driving tunnels of various diameters in different kinds of water-bearing ground by the freezing-process. The information that has been obtained is very complete, and as general in nature as it could be made.

After carefully watching this test, and studying the results obtained, I am of the opinion that freezing may be made a very valuable auxiliary to tunneling in water-bearing sands and gravels. It will not, as expected by some, supersede the shield and compressed-air methods, nor will it, in all cases, avert the necessity for the use of compressed-air; but it will, if intelligently applied, eliminate many of the uncertainties and difficulties of the methods now in use. Figs. 17 and 18 are views of the pilot-tunnel during the test.

Tunnels in Construction (Fig. 1).—New York & Jersey Railroad Co.'s tunnels: Of the tunnels at present under construction, the first to be commenced was the old Hudson tunnel. This work was commenced in 1879 by D. C. Haskin, who sunk

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a shaft at the foot of Fifteenth street, Jersey City, and drove about 1,542 ft. of the north tunnel and 600 ft. of the south tunnel, chiefly by the pilot-tunnel method above described. A shaft was also sunk on West street, at the foot of Morton street, on the New York side, and a short length of the north tunnel was driven towards the Jersey shore. In 1882 the work was temporarily abandoned, to be taken up again in 1890 by S. Pearson & Son, who continued work in the north tunnel from Jersey City by the same method that Haskin used until perhaps 100 ft. had been driven. A shield was then built and an iron-lined tunnel constructed of 1,900 ft. in length, making the total length of the tunnel about 4,000 ft. by August, 1891. At this time, the funds being exhausted, the work was abandoned.

The undertaking was resumed by the New York & Jersey Railroad Co., with Mr. William McAdoo as President and Mr. Chas. M. Jacobs as Chief Engineer, in 1901, and now forms part of an extensive system. This consists of twin tunnels from the station of the Delaware, Lackawanna & Western Railroad Co. in Hoboken, past the Erie Railroad station and the Pennsylvania Railroad station to the Jersey City terminal of the Central Railroad of New Jersey. These tunnels are connected with the New York side by the Hudson & Manhattan Railroad twin tunnels running from the Pennsylvania Railroad terminal to Cortlandt street, where a large terminal is being built; and by the twin tunnels (the original Hudson tunnel) running from the foot of Fifteenth street, Jersey City, to the foot of Morton street, and thence up Sixth avenue to 33d street, where a terminal will be built. The total length of single track tunnel in this system is about 12 miles.

The tunnels of the Pennsylvania Railroad Co. were commenced during the administration of the late A. J. Cassatt, with Mr. Samuel Rea in charge, and are being constructed under the general supervision of a Board of Engineers appointed by the late Mr. Cassatt, with General Raymond as Chairman, and with Mr. Alfred Noble and Mr. Charles M. Jacobs as Chief Engineers of the East and North River divisions, respectively. There are two tunnels, 25 ft. 10 in. wide and 21 ft. 5 in. high, which start from the portal at the west side of Bergen hill on the Hackensack meadows through the hill to a point

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224 ft. east of the Weehawken shaft. For the most part these tunnels are in trap rock and driven by ordinary methods, but the contact of the trap and the sandstone occurring in the shaft at Weehawken, the rock just east of the Weehawken shaft is mixed in character. From the shield-chambers, 224 ft. east of the Weehawken shaft, the tunnel is first in rock, then in rock overlain by sand and gravel, then sand and gravel, and then silt, which extends across the whole width of the river. The tunnels were driven by shield and compressed-air methods, as described before. While passing through the sand and gravel it was found necessary to drive one tunnel at a time, the other being used to drain the ground somewhat. On the New York side the rock-level rises again, and the tunnel, from being entirely in silt, is entirely in rock, and the iron-lined tunnels end at a point about 1,070 ft. east of the bulkhead. Where the iron-lined tunnel was entirely in rock, it was driven ahead of the shield, and a concrete cradle laid to receive the iron lining, which was erected behind the shield as it advanced.

From the shield-chamber to the east are twin tunnels that extend to the terminal station, which pass through rock varying in character from mica schist to hard trap, and overlain by sand and gravel. Where wholly in rock with good rock cover, these tunnels are being excavated by ordinary rock-tunneling methods; where partly in rock and partly in gravel, they are excavated by cut-and-cover methods. From the terminal station eastward run two three-track tunnels, one under 32d and one under 33d street. Just west of Fifth avenue the construction is changed to two pairs of single-track tunnels, which run into the shafts near First avenue. These tunnels are chiefly in rock, but there are short lengths where the rock cover is very thin, and in places it runs out altogether and there is part of a face of sand and gravel. The construction is again changed at the shafts near First avenue, where four iron-lined river-tunnels start and pass through rock, rock and sand and gravel, into a full face of quicksand, thence through a reef of rock into sand again, then back into rock and through the river-shaft in Long Island City for about 2,000 ft. to East avenue. From East avenue to Thompson avenue the work is either cut-and-cover or open cut. The total length of single tunnel in this system is about 18 miles.

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With reference to the material through which these tunnels have been or are being driven, the prevailing materials forming the beds of the North and East rivers, while apparently quite different, have one or two points of striking similarity. They are composed of extremely fine particles, and both in their natural states contain surprisingly large percentages of water.

The silt of the North river resembles a soft clay, and contains at the level of the tunnels about 33 per cent. by weight of water, which is considerably more than sufficient to fill the voids, if the particles were spheres. It is almost impervious to air and water, and flows freely under pressure. It apparently will not stand at any angle.

The chief material found in the East river is an extremely fine red sand, divided by occasional horizontal streaks of stiff red clay. It contains about 22 per cent. by weight of water and is a typical quicksand. Under the action of a sufficient pressure of compressed-air, the water is expelled from it and it will stand vertically or even overhanging, and when in this condition it resembles a rotten sandstone. It permits the escape of compressed-air, though not at first freely, and when the pressure of air falls below the balancing-pressure it becomes quite liquid.

At one time it was thought that the tunnels under the North river would have to be supported against downward displacement, which it was supposed the vibrations caused by traffic might cause. A very careful investigation of the whole matter was made by General Raymond, who studied all the reliable evidence obtainable from the old Hudson tunnel and from the Pennsylvania tunnels, and made a long series of observations of the pressure exerted on the tunnels in both cases by the silt by means of a specially constructed pressure-gauge. He points out that, provided it be made tight against water and silt, it is not possible for a tunnel that is lighter than the displaced material to settle, and that the tunnels of the Pennsylvania Co., which, while normally lighter than the silt, may, when a train passes through, be momentarily heavier, will have to be held against both upward and downward motion, in order that they might be perfectly safe under all conditions, and that the only effect, if any, of vibrations, will be to tend to cause the tunnel to move in the direction of the resultant of the pressures, which, in general, is upwards.

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The tunnels of the Rapid Transit Commission, now under construction, consist of iron-lined twin tunnels in continuation of the Subway, from the Battery under the river to Joralemon street, Brooklyn, thence to a connection with the Fulton street line, Brooklyn. The ground passed through, on the Manhattan side, was rock of a variable kind, some hard, and some soft and decomposed with irregularities and fissures; the roof of the excavation was timbered, and the iron lining was erected by means of an erector that traveled on rollers attached to the lining, the space between the rock and the lining being packed with dry rock and grouted; from this rock the tunnel ran into sand and gravel and then again into rock; on the Brooklyn side the work was entirely in sand and gravel and carried on entirely with shields.

The tunnels of the New York & Long Island Railroad Co. are twin iron-lined tunnels of about 15 ft. 3 in. diameter driven in rock from 42d street to Man-o-war's Reef and thence partly in sand to Long Island City. From New York to Man-o-war's Reef it was expected to avoid compressed-air work, as the tunnel is entirely in rock, but fissures occur in the rock and compressed-air has been used to some extent.

All the tunnels above mentioned are under construction at present, and make a total of 38 miles of underground workings.

In addition to these works under construction it may be mentioned that about 66 miles of new subways are projected, and most of them must be constructed in the near future.

As regards the illustrations of this paper, I am grateful for permission to reproduce Fig. 2 from a paper by Mr. William Sooysmith entitled the Hudson River Tunnel,[1] and Figs. 10, 11, 12 and 13 from the working-drawings of Messrs. S. Pearson & Son, contractors for the East River tunnels of the Pennsylvania Railroad Co., and the photographs from the collection of the Pennsylvania, New York & Long Island Railroad Co.

[1] *Transactions of the American Society of Civil Engineers*, vol. xi., pp. 314 to 323 (1882).

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The Verschoyle Pocket Transit.

BY W. DENHAM VERSCHOYLE, BALLISODARE, IRELAND.

(New York Meeting, April, 1907.)

In designing a pocket instrument whereby any given horizontal or vertical angle may be closely approximated, the following points should be kept in view, if general utility is aimed at:

1. The instrument should be light and compact, so that it can be carried in the pocket, or in a sling.

2. It should be capable of observing any angle whatever, horizontal or vertical.

3. Since time is often an important consideration in making a preliminary survey, it should be capable of giving the horizontal and vertical angle at one observation.

I do not know of any instrument that will fulfill these requirements, except the one here described.

Where the depression- or elevation-angle of the distant object, whose bearing is required, exceeds 50 or 60 degrees, it becomes difficult or impossible with a prismatic compass, for instance, to obtain accurate results; and since it frequently happens, particularly in mining, that much work requires to be done beyond that limit, it is obvious that an instrument which can be used to observe a high angle just as easily as a low one, if not absolutely a necessity, will at least be a great convenience.

How this "pocket transit" attains to the required degree of universality will be understood from the diagram, Fig. 1, in which B is a beveled, graduated circle, attached to and freely revolving about the center of the magnet A. Light falling on B is reflected through a window in F (the compass-box), along the line AC, and is again reflected back to the eye at D by the prism C.

By a simple arrangement, the sight-line DCE can be completely revolved in the vertical plane DE, which is normal to AC. It is clear, then, that no matter through what vertical

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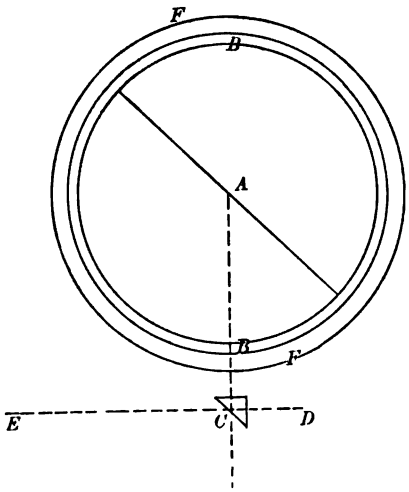

FIG. 1.—DIAGRAM, ILLUSTRATING THE PRINCIPLE OF THE POCKET TRANSIT.

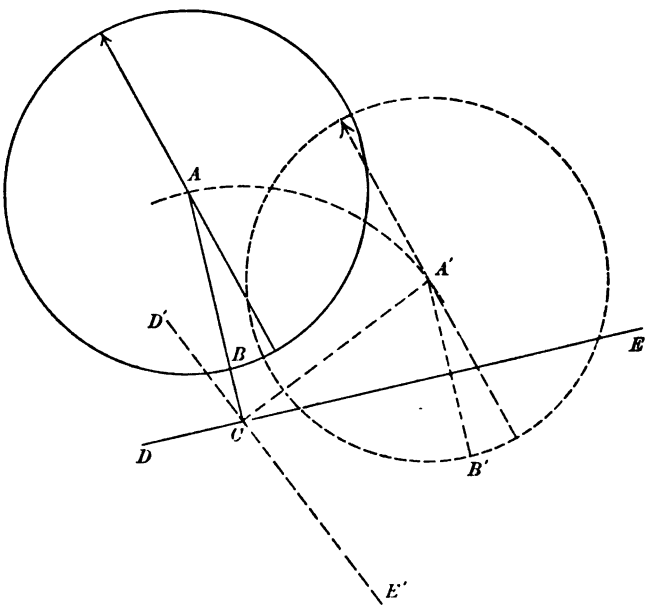

FIG. 2.—DIAGRAM, ILLUSTRATING THE PRINCIPLE OF THE DOUBLE OBSERVATIONS MADE WITH THE POCKET TRANSIT.

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FIG. 3.—APPEARANCE OF THE VERSCHOYLE TRANSIT WHEN CLOSED.

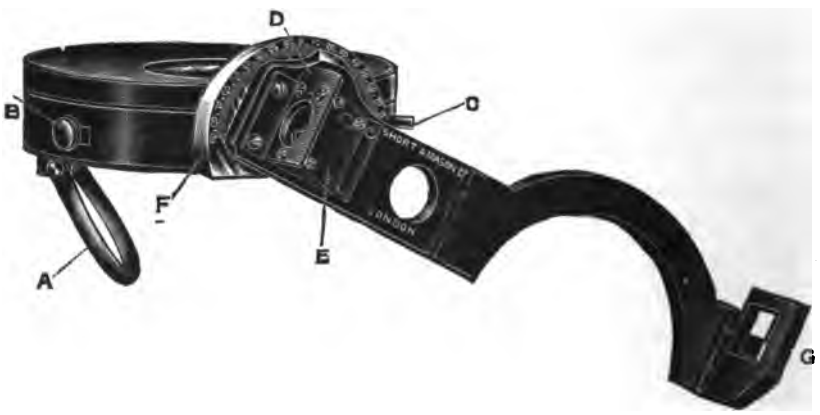

FIG. 4.—THE VERSCHOYLE TRANSIT, WITH SIGHT-ARM DEPRESSED.

A, thumb-loop; B, needle-lifter; C, needle-brake; D, back-sight hole; E, prism-box; F, vertical arc; G, fore-sight.

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angle DE has been rotated, the figures at B can be read with equal ease, by means of the prism C. Also, by placing on the arm to which are affixed the prism C, and the fore- and back-sights defining the line DE, an alidade which traverses a vertical graduated circle affixed to F, and by having a small spirit-level affixed to F, the bubble of which is visible also at D, we are able, in observing the magnetic bearing of any distant object, to secure an automatic registration of the vertical angle by bringing the bubble to the center.

To obtain freedom for the sight-line in the vertical plane, it was necessary to make it tangential to the horizontal circle.

FIG. 5.—THE VERSCHOYLE TRANSIT, WITH ARM ELEVATED.

If, then, A were used as the center of the instrument, there would be an error due to eccentricity, in observing horizontal angles.

If, however, C be taken as center, and the instrument be considered in any two positions, the following will be true in all cases:

Let $ABCDE$, the plain-line part of Fig. 2, represent any position of the instrument. Let the sight-line DE be moved through any horizontal angle ECE', to the second position, as shown by the dotted line, and take B', the same point on the scale as is cut by AB. Then $A'B'$ is parallel to AB, and the angle $ECE' = ACA'$, and therefore $= CA'B'$ (Eucl. I., 29),

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which means that any horizontal angle through which the sight-line is rotated, with C as center, is equal to the angle through which AB passes, in the same motion, with A as center.

The instrument can be used as a clinometer and grading-level; and a tripod of special design is supplied if desired. To use it as a clinometer, the bottom side of the arm is laid on the surface the dip of which is to be measured, and the compass-box is slightly revolved until the small bubble, viewed through the window at the top of the box, is brought to the center of the spirit-level, when the angle may be read off the vertical arc.

Figs. 3, 4 and 5 show the general appearance and construction of the instrument. It is manufactured in the United States by the Taylor Brothers Co., Rochester, N. Y.

Biographical Notice of William George Neilson.

BY JOHN BIRKINBINE, PHILADELPHIA, PA.

(New York Meeting, April, 1907.)

MR. NEILSON was born Aug. 12, 1842, at Philadelphia, Pa., where he died Dec. 30, 1906. His business career began with his graduation, in the class of 1862, from the Polytechnic College of the State of Pennsylvania.

For a short time after graduation he was instructor in mathematics at the college, and then became associated with Messrs. Booth & Garrett, chemists, of his native city. In 1867 he was sent to the Adirondack mountains to care for mining interests of the late Jay Cooke, and spent about three years there, operating the forges at Elizabethtown, Essex county, N. Y. He also prepared an interesting monograph on Catalan forges and their operations.

In 1868 he published a chart showing the material progress of the United States, and the cost of certain staple products during a series of years, which is believed to be the first of its kind issued in this country. In 1870 he made a trip to the Pacific Coast, prior to entering the service of the Pennsylvania Steel Company.

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In 1871, when the Logan Iron & Steel Co., of Burnham, Pa., failed, Mr. Neilson was appointed receiver, reorganized the company, became its General Manager, and spent more than four years in this work. In the meantime, he became interested in the Freedom forge of the same place, and, with Mr. Wm. Burnham and Mr. Milton Brosius, laid the foundations for what is now the Standard Steel Works. From Jan. 1, 1877, to April 1, 1890, he was General Manager of these works, and closely associated with the Baldwin Locomotive Works. In 1878 he was in charge of the first consignment of Baldwin locomotives to Europe, which, with a picked crew of men from Baldwin's, he took to Eydtkennen, Russia, and placed in service.

In 1890 he resigned from the Standard Steel Works and accepted a position in connection with the Chester Rolling Mills; later, he became and remained until 1893 Vice-President of the Wellman Iron & Steel Co. From 1893 to the end of 1895 he was General Manager of the Taylor Iron & Steel Co., of High Bridge, N. J.

For the past eight years Mr. Neilson had been the Treasurer of the Keystone Drop Forge Works, of Chester, Pa.

While Mr. Neilson was with the Standard Steel Works he became interested in a large tract, at the head of Keene valley in the Adirondacks, which was about to be sold for its timber. Forming a company, chiefly of Philadelphia and New York men, he purchased this property and established the Adirondack Mountain Reserve, of which he served as President for 17 years, and which tract remains to-day one of the most attractive forest-preserves in the Adirondack region. For many years he was a member of the Pennsylvania Forestry Association. Mr. Neilson was a pioneer in mining bauxite, and for a number of years had been practically engaged in the management of Southern mines from which the mineral is obtained.

He was also for many years a Director of the Union Insurance Co. of Philadelphia, and was actively interested in the Young Men's Christian Association of Philadelphia, serving as President during the years 1899, 1900, and 1901, and as Director continuously, after Jan. 1, 1888, until the time of his death.

Mr. Neilson's connection with the Institute began in 1872, the year after its organization, and continued until his death. From the beginning, he was loyal and enthusiastic in his sup-

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he Institute. In 1876 he was selected as S
tennial Committee," which collected and ad
me of the Institute, a special fund of a little
n the maintenance of headquarters, for the entertain-
l information of visiting engineers, at the Centennial
on in Philadelphia. At these headquarters a profes-
versazione was held weekly, and an addres s was made
distinguished expert. Moreover, the room s were kept
usly open, and provided with guide-book and much
propriate literature and maps, while the S ecretary and
ants were always prepared to assist with advice, intro-
etc., visiting colleagues engaged in professional in-
, inside or outside of the Exposition. Thus, there were
l to foreign guests 493 written letters (besides innu-
cards) of introduction, and 43 detailed schedules of
accompanied with maps of the United was intrusted
e routes of these journeys were laid do with the rela-
anagement of this difficult undertaking those of his
eilson; and the results he accomplished his executive
all sum at his disposal surprised eve ere gratefully
who had had reason already to rely upo he weekly re-
The headquarters under his charge enjoyed. For-
hundreds of visiting engineers, and elty and useful-
ere numerously attended and keenl
sts were greatly impressed with the no
vell as the cordiality of this form of p
l foreign technical literature was l
dgment of it. Among the results
on was the donation to the Ameri
Engineers of mineral and metallurgi
, Sweden, Russia, Spain, Portugal
, England, the Australian colonies,
and the United States, amounting in estimated value
$250,000.[1] The Emperor of Germany, in recognition
rvices rendered by Mr. Neilson to visiting German en-
decorated him with the Order of the Red Eagle, fourth

ort of the Centennial Committee, *Trans.*, v., 31 (1876–7). These col-
re first left with the Pennsylvania Museum and School of Industrial
bsequently (1882) transferred to the National Museum at Washington,
Trans., x., 243 (1881–2).

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The report of the Centennial Committee, cited above, reads as follows:

"The Committee cannot close its report without thanking its Secretary, Mr. William G. Neilson, for the energy, ability and fidelity with which he discharged his onerous duties; and also Mr. Edward Nichols, who, from April 1 to Dec. 1, 1876, was the Assistant Secretary, and ably assisted Mr. Neilson in all his duties."

In this connection it is interesting to observe that the only contribution made by Mr. Neilson to the *Transactions* was an appreciative Biographical Notice of Mr. Nichols,[2] who died in 1892.

It is not too much to say that this intelligent and well-organized enterprise of professional and international courtesy did more to win recognition for the Institute, throughout the world, than any other single event in its history, and that the credit for this happy result was chiefly due, not to the Committee which suggested the plan, but to Mr. Neilson, who successfully executed it.

On Jan. 2, 1907, numerous friends gathered at the funeral service and bade a sorrowful farewell to one who had been loved and esteemed by all present, and whose memory will be cherished by many friends in and beyond the American Institute of Mining Engineers. We saw in the casket the peaceful expression which had been familiar to us for many years; except for the gray hair, it seemed to me to be the same expression of friendly recognition which characterized him when, as college students, we first met. It bore no sign of suffering. Death called him at his office desk on the last Saturday of the year 1906, and as he closed his accounts with his business he passed from earth.

As a collegian and alumnus, highly esteemed; as a friend, beloved; as a business man, respected and trusted; as a Christian, earnest; as the head of a family, affectionate and wise; in all things faithful, Mr. Neilson had a host of friends which his home could not accommodate as mourners. Nor are the mourners confined to our own country; for in his European journeys, and in the service which he rendered to the Institute, as above described, he made many friends abroad whose sorrow at his death echoes that of his own countrymen.

[2] *Trans.*, xxi., 76 (1892–3).

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Biographical Notice of Thomas Septimus Austin.

BY ARTHUR S. DWIGHT, NEW YORK, N. Y.

(New York Meeting, February, 1908.)

THE professional career of Thomas Septimus Austin, who died at El Paso, Tex., Aug. 23, 1906, was contemporaneous with the growth of the silver-lead smelting-industry of the Far West, to which his talents and zeal contributed in no small degree.

Born at Stratford, Conn., Dec. 7, 1853, he was the seventh son in a family of thirteen. His parents were well-to-do; and he received a thorough preliminary training at the Hopkins Grammar School, New Haven, Conn., passing thence to the Columbia School of Mines, in New York, where he was graduated in 1876.

His first professional work was that of analytical chemist with a Cuban sugar company. In this position he spent a year, and, whatever else of benefit he may have acquired during that year in Cuba, he gained a facility of speech in the Spanish language which was of inestimable value in his subsequent work in Mexico and the southwestern United States.

In 1877, he went to the Rocky Mountains, and, under the firm name of Murphy & Austin, opened an assay-office in Leadville, Colo., which was just then coming into public notice as a rich silver-lead camp. The next year, however, he entered the service of the Germania Smelting & Refining Co. at Salt Lake City, Utah, where he remained until 1882, first as assayer and chemist, and later as Assistant Superintendent. The Germania bears the honorable distinction of being one of the first of the large "custom" lead-smelters erected in the West. Its founders may be called the pioneers of the modern lead-smelting practice, and the Germania served, to a large extent, as a model for the many plants that sprang into existence during the next few years in Utah, southern Idaho, Colorado and other lead-producing districts. The formulation, for the first

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time, of a logical and practically satisfactory working-theory of slags for the lead blast-furnace was a distinct advance, and constituted the keynote of this successful practice. In fact, the consistent pursuit of this practice became the distinguishing mark of that school of metallurgists, headed by Anton Eilers, August Raht and O. H. Hahn, and including many younger engineers trained by them, which has kept its identity clearly defined through the succeeding thirty-odd years of expansion and industrial change. Under the influence of this group of distinguished metallurgists, Mr. Austin was trained; and much of his thorough grasp of detail, and the sound metallurgical judgment which he displayed in his later work, must be credited to the invaluable discipline, and the correct molding of his fundamental professional habits, which he gained in this period of his career.

In 1882 he took charge of a newly erected silver-lead smelting-works at Ketchum, in the Wood River district of Idaho; but after a year he was obliged to relinquish this post and return to Salt Lake City, by reason of illness from lead-poisoning, which he had there contracted. In 1883 he again became connected with the Germania smelter, this time as Superintendent, and remained there until 1887, when he accepted the position of Superintendent of the Rio Grande Smelting Works at Socorro, N. M., newly built by the late Gustav Billing, and became an important contributor to its success. This establishment, under the management of Albert F. Schneider, a classmate and, at the Germania, a technical associate of Mr. Austin, enjoyed an exceptionally favorable basis in its supply of lead-ores, at first from the famous Kelley mine, near Socorro, and later from the mines of old Mexico. But the U. S. Treasury ruling of 1890, practically prohibiting the importation of lead-ores from Mexico, crippled the operations of the Socorro plant, and robbed it of its strategic importance, so that in 1891 it was obliged to suspend operations entirely.

On leaving Socorro, Mr. Austin made an engagement with Messrs. M. Guggenheim's Sons, who were about to embark in the construction of a new smelting-plant at Monterey, Mexico. This undertaking was beset with many serious difficulties, such as might be expected in the inauguration of a large technical industry in a new country. With his usual devotion to his

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work, Austin did not spare himself; and his constitution suffered severely, and perhaps permanently, from the physical strain and unsuitable conditions of living thus incurred.

The first furnace of this plant was started in January, 1892. By the end of March, 1892, he had five, and by March, 1893, ten, furnaces in operation; but in September of the latter year his contract expired, and he was glad to give up the work in order to take much-needed rest and recuperation. The following period of leisure he devoted with characteristic zeal to a course in experimental electricity at Columbia University, being greatly impressed with the value of such knowledge in its bearing on the future development of metallurgy.

Early in 1894 he returned to the West, and engaged in general mining work in New Mexico, and later undertook to rehabilitate at Chihuahua, Mexico, a mining- and smelting-plant, which he operated successfully for a time. This enterprise was suddenly terminated, about the end of 1895, by a serious caving-in of the mine upon which it depended.

In August, 1897, the superintendency of the El Paso Smelting Works at El Paso, Tex., then a branch of the Consolidated Kansas City Smelting & Refining Co., was offered him by the writer, who, as General Superintendent of the company, had been at work for several months on a serious problem of reorganization. This plant enjoyed an unusually good strategic position, with an assured supply of suitable ores from the company's own mines and ore-buying agencies in Mexico and the Southwest. It had, in fact, everything needed for a highly profitable business, except economical operation. Its difficulties in this respect were complicated and deep-seated, but had at last been correctly located, and a plan of readjustment had been inaugurated, which finally brought success. This scheme, besides calling for the solution of many perplexing technical problems, required the gradual remodeling of the plant, and the building up, by patient and intelligent discipline, of a reliable force of trained Mexican furnace-men from the irresponsible class peculiar to the border country. It was the very task for which Austin was fitted by temperament and training; and the company was fortunate indeed in securing his services. For the next nine years, constituting the remainder of his busy life, he was engaged in the continuous

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prosecution of this work, with a signal success which brought ever-broadening usefulness and responsibility. To his metallurgical skill, he added that wonderful constructive patience which is content to weave, thread by thread, the fabric of success. Such a combination is unconquerable.

His men caught his spirit and transmuted it into results. The El Paso plant soon came to be noted for the excellence of its metallurgical practice and the efficiency of its organization, while the earnings showed corresponding improvement, so that the works took rank among the most profitable in the country.

By the great merger of smelting interests accomplished in 1899, the El Paso plant passed under the ownership of the American Smelting & Refining Co., but without disturbing Austin's work. In July, 1901, during his temporary absence from El Paso, the blast-furnace department of his plant was completely destroyed by fire. This disaster was a blessing in disguise, since it enabled him to put into practice some of his well-digested theories regarding furnace-construction and the handling of materials. The system of mechanical charging of the blast-furnaces which he then inaugurated and perfected has proved highly successful in practice. The new power-plant of modern design greatly cheapened the cost of motive-power, and rendered possible many economies throughout the works. By December, 1902, the furnaces were again in operation; and the El Paso plant stands to-day as one of the most economical and thoroughly satisfactory lead-smelting establishments on this continent. In its recognized excellence of design and detail, its efficient technical organization and its well-balanced practice, it presents an honorable and indelible record of Austin's professional skill and unwearied industry.

It was natural that such ripe experience and judgment as his should be called into consultation when technical difficulties arose at other plants of the American Smelting & Refining Co.; and thus the scope of his duties was gradually enlarged until, in 1903, he was appointed General Superintendent of the Southern Department of that company, including, through the recent acquisition of the Guggenheim interests, sundry silver-lead plants in Mexico, as well as that of El Paso. Besides his purely technical work in this capacity, he had a voice in the

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settlement of commercial questions, concerning which his judgment was always helpful in council.

In the summer of 1906, his health, undermined by his unremitting work, suddenly failed. Accepting too late this warning, he was, in August, just starting on a fishing-trip for rest and relaxation, when he was stricken with diabetes, and died after an acute illness of but two days.

His death was not only a great loss to his profession, but also a sorrow and shock to a large circle of professional friends and business associates, who had long esteemed him as a man of rare character and abilities. Most of all was he mourned by the many young engineers whom he had trained in his art, and whose affection he had won by personal kindness and painstaking interest, while his knowledge and skill had commanded their respect. His workmen also honored and loved him; for all of them, from the American foreman to the humblest Mexican laborer, had always found him a sympathetic listener and a just arbitrator of any real grievance. Behind his quiet, unobtrusive manner there was a great reserve force which all instinctively recognized, and to which all responded with their best service. And thus he was more than an ordinary "good judge of men," since he saw in them not only what they actually were, but also what they had the capacity to become, under the stimulus and guidance which he could supply. This is a higher and rarer gift than the mere mechanical judgment, however keen and accurate, which weighs a man just as he is, without discerning or inquiring whether he can grow. And this higher gift not only serves immediate business interests: it wins the love and loyalty of men—which is, in fact, a business asset of no small value.

Mr. Austin's engrossing duties as an executive or adviser doubtless prevented him, as so many other brilliant men have been prevented in similar circumstances, from systematic study in his own specialties, and from making such a record of his observations as might hand down to his successors the knowledge he had gained. He remained to the end a diligent student, and, moreover, he was able to state his views and conclusions with clearness and force. It is a great pity that he never found time to do this, in logical and comprehensive form, through contributions to current technical literature.

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Fortunately, he committed to writing, in the form of private letters, many of his technical results and opinions; and some of these have been edited and published, since his death, in the *Mining and Scientific Press* of San Francisco, by his brother, Prof. L. S. Austin, of the Michigan College of Mines. Although these articles lack the consecutive, logical and comprehensive character which they would have possessed if originally prepared for publication by T. S. Austin himself, they contain many valuable facts and suggestions concerning the art of lead-smelting.

Mr. Austin married in June, 1890, Miss Dorothy Lockhart of Albuquerque, N. M., who, with one daughter, survives him. His domestic life was happy, and afforded him great solace and stimulus.

He became a member of the Institute in 1883.

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Blow-Holes in Steel Ingots.

BY E. VON MALTITZ, METALLURGICAL ENGINEER, SOUTH CHICAGO, ILL.

(Toronto Meeting, July, 1907.)

In his highly interesting paper, Piping and Segregation in Steel Ingots,[1] Prof. Howe emphasizes the effect of successive phases of internal pressure in the ingot in the evolution of gas, and the consequent formation of blow-holes. From his statements in other places, it may be inferred that he does not here intend to ignore the other variable factor, temperature, but rather to discuss this one more particularly. At all events, it is my purpose not to engage in controversy with Prof. Howe, but rather to contribute, towards the complete elucidation of this important subject which we all desire, my own views and experience. If, in doing so, I specially emphasize other factors than pressure, I need not be understood as denying what I do not think it necessary to discuss fully. Nor, in presenting my own original conclusions, need I be understood as claiming to be the first who ever reached or announced them. Many things are asserted or suggested upon inconclusive evidence; and to confirm such tentative propositions may be as valuable a service as to overthrow them.

In the following passage, Prof. Howe adduces analogies, based upon the behavior of charcoal and of water, respectively, which seem to me questionable:

> "In general, the solvent power falls as the pressure falls; and in general it rises as the temperature falls. Thus, to heat a solid, for instance charcoal, may expel part of its dissolved gas; and a tumbler of water drawn cold from the faucet gradually evolves gas, as it stands and warms up on the sideboard." (p. 72.)

Charcoal is a solid body, which has the property not only of absorbing gases in large quantities, but also of expelling the thus absorbed gases upon heating. A comparison between charcoal and iron as to the absorption of gases is scientifically not possible; the gas-content of charcoal is partly due to the

[1] This volume, pp. 3 to 108.

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incomplete decomposition of the wood, partly to the extremely great porosity of the charcoal, which gives a very large surface. This surface attracts certain gases, and favors formation of new gases in the cold by the contact with the oxygen and moisture of the atmosphere.[2] To arrive at a right comparison we must take solid iron and see if this body is able to take up gases and to set them free again with changing temperature.

I. Does Solid Iron Absorb and Release Gases with Changing Temperature?

Graham (1866)[3] found that iron together with other metals has the property of absorbing hydrogen; Parry, Troost and Hautefeuille verified this discovery. The latter experimenters heated pieces of solid iron, of 500 g. each, up to 800° C. in an atmosphere of hydrogen and of carbon monoxide, respectively. They found that the iron at this temperature was able to absorb three times as much hydrogen as carbon monoxide; these gases were again given off *in vacuo*. This points to the fact that solid iron is able to absorb gases upon heating. A. Ledebur[4] found that steel became very brittle, when pickled in dilute sulphuric acid or hydrochloric acid. An analysis of such steel showed that it had taken up 0.002 per cent. of hydrogen. Johnson,[5] Hughes,[6] and Baedecker,[7] experimenting along the same line, came to the same conclusions. H. von Jüptner[8] found that hydrogen is able to diffuse through sheet iron, when corroded by dilute sulphuric acid or hydrochloric acid, and produces large bubbles in defective places, in which bubbles the gas is contained under considerable pressure. Roberts-Austen[9] found that pure iron, obtained by electrolytic precipitation from a chemically pure solution of ferric chloride, gave up large quantities of hydrogen, when heated in water up to 70° C., and that it continued to do so,

[2] *Poggendorf's Annalen*, vol. cix., p. 353 (1860).

[3] *Graham, Collected Works*, p. 279.

[4] *Stahl und Eisen*, vol. vii., p. 681 (1887); vol. ix., p. 745 (1889).

[5] *Proceedings of the Royal Society*, vol. xxiii., p. 168 (1875).

[6] *Journal of the Society of Telegraph Engineers*, vol. ix., p. 163 (1880).

[7] *Zeitschrift des Vereines deutcher Ingenieure*, vol. xxxii., p. 186 (1888).

[8] *Siderology: The Science of Iron* (Translation by Charles Salter of *Grundzüge der Siderologie*, by H. von Jüptner) vol. i., p. 285 (1902).

[9] *Proceedings of the Institution of Mechanical Engineers*, 1891, p. 543. Report to the Alloys Research Committee.

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when heated *in vacuo* in a porcelain tube, up to 1,300° C., at which temperature the liberation of hydrogen became a minimum. Repeated heating of the same iron up to 1,300° C. deprived it of all its hydrogen; it could, however, be loaded again with hydrogen, when the iron was hung as the negative electrode in dilute acid. All these investigations point out that iron is able to absorb hydrogen at low temperature, and that its maximum solvent power for this gas must be close to 70° C.

E. Heyn [10] heated soft steel in an atmosphere of hydrogen up to 1,000° C. In order to keep the hydrogen in the steel he quenched the test in cold water, and found that it exhibited great brittleness, decidedly more than similar steel when heated up to 1,000° C. in air and then quenched in cold water. Analyzing the steel, he found that it contained 0.00019 per cent. hydrogen. After heating the steel in hydrogen up to 1,000° C. he cooled it down slowly in air. When this cooling was continued until the temperature of the steel was below 730° C. any noticeable effect on the steel by this treatment could not be detected.

From this it follows, that iron (and steel) must absorb hydrogen at temperatures between 730° and 1,000° C., and further, that iron (and steel) must experience a decrease in its solvent power for hydrogen, when cooling below 730° C., or, in other words, the metal must liberate hydrogen when cooling below 730° C.

From these investigations we arrive at the following conclusions:

(*a*) Solid iron (and steel) has two maxima in its solvent power for hydrogen; one at about 70° C. and another between 730° and 1,000° C.

(*b*) Solid iron (and steel) gives off hydrogen, absorbed at a temperature above 730° C., when cooling below this temperature.

(*c*) Solid iron (and steel) gives off hydrogen and other gases upon heating.

Water, which furnishes the second analogy, is a liquid body, and should therefore be compared with liquid bodies only—in this case with molten steel. We know that water sets free

[10] *Stahl und Eisen*, vol. xx., p. 36 (1900).

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occluded gases upon heating, and that continued boiling will drive out all the air contained in it. On the other hand, however, we know, also, that water sets free occluded gases upon freezing, and that these gases, when they cannot escape, form blow-holes in the block of ice. Water, made gas-free by continued boiling, will freeze without blow-holes; but if we blow air through such gas-free water, as it approaches the freezing-point, the ice, when formed, will have blow-holes.

The liquid water has therefore two points of minimum solvent power for gases, the freezing- and the boiling-point. Between these two minima, there must naturally lie a maximum of this solvent power—at what temperature I am unable to say. But it suffices for our purpose that water, when fluid, is able to absorb gases.

II. Is Molten Steel Able to Absorb Gases?

We know that steel, when freezing, evolves gases and forms blow-holes, exactly like freezing water. And, like water, the steel must have had, at some point during the time it was in the molten state, the capacity to absorb gases in such amount that, upon freezing, it is unable to keep them in bond, and therefore sets them free. In other words, solid steel cannot contain as much occluded gas as molten steel. When we melt this solid steel in the open-hearth furnace, it must at some temperature acquire a decidedly greater solvent power for gases; otherwise, no blow-holes would be formed.

If water is able to absorb gases at a certain temperature, which is above its freezing-point, why may not steel do the same with regard to hydrogen and nitrogen?

Are we not forced to accept a third maximum in the solvent power of steel for gases, which lies away above 1,000° C., in fact, at a considerable distance from the melting-point?

Heating to still higher temperatures must decrease the solvent power for gases rapidly again, for we know that steel, poured extremely hot (and without overoxidation), will give perfectly solid ingots and castings.

To compare water once more with molten steel, we find that a solid block of ice or steel, containing a minimum of gases in the form of blow-holes, will, as it is melting and being subjected to a continuous rising temperature, approach a maximum in its

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solvent power for gases, and after having passed this maximum will, upon further heating, experience a rapid decrease in its solvent power.

If we accept the theory that increasing temperature lowers the solvent power of molten steel for gases, why is it that the steel does not lose all its gases in the furnace, where it is at its highest temperature? And if, with decreasing pressure, the solvent power also falls, why are not the gases driven out in the furnace, where there is a slight pressure at the incoming side for gas and air, but a relative depression at the outgoing side, due to the draft? Each reversal of the gas and air puts more than 50 per cent. of the bath under lower pressure, and its solvent power should fall with that decrease. The opportunity to give off gases would certainly be splendid here, on account of both higher temperature and lower pressure. But does the steel show such action? Why does it wait with the liberation of the occluded gases until it is poured into the molds? Are we not justified in looking for other reasons for the formation of blow-holes in steel ingots than those given by Prof. Howe? To arrive at the right conclusions, it is necessary to first study the nature of the gases which are found in blow-holes, and then to ask if these gases, which are found in these cavities, are alone responsible for their formation—in other words: if the blow-holes are formed solely by those gases found inclosed in them, without the interference of a third gas.

III. What is the Nature of the Gases Found in Blow-Holes?

The classical investigations of Dr. F. C. Müller, Messrs. Stead and Richards, and others, on the gases contained in blow-holes of steel ingots, have shown that these gases consist chiefly of hydrogen and nitrogen. A little carbon monoxide was found in a few instances, but always in too small a percentage to make it one of the chief constituents.

For illustration I give a few analyses of blow-hole gases:

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TABLE I.—*Analyses of Blow-Hole Gases.* Taken from *Stahl und Eisen* for 1882 and 1883.

No.	Volume of Gases.	Composition of Collected Gases. H_2.	N_2.	CO.	Authority.
	Per Cent.	Per Cent.	Per Cent.	Per Cent.	
1	16.5	68.8	30.5	0.0	Dr. F. C. Müller.
2	51.0	78.1	20.7	0.9	Dr. F. C. Müller.
3	17.0	92.4	5.9	1.4	Dr. F. C. Müller.
4	36.0	85.4	14.3	0.6	Dr. F. C. Müller.
5	20.0	64.5	35.4	0.0	Dr. F. C. Müller.
6	22.0	86.4	12.7	0.4	Dr. F. C. Müller.
7	6.0	54.7	45.3	0.0	Dr. F. C. Müller.
8	48.0	90.3	9.7	0.0	Dr. F. C. Müller.
9	45.0	77.0	23.0	0.0	Dr. F. C. Müller.
10	29.0	76.7	26.3	0.0	Dr. F. C. Müller.
11	21.0	81.3	18.1	0.0	Dr. F. C. Müller.
12	25.0	67.0	30.8	2.2	Dr. F. C. Müller.
13	60.0	88.8	10.5	0.7	Dr. F. C. Müller.
14	44.0	80.4	17.9	1.3	Dr. F. C. Müller.
15	21.0	67.1	33.3	1.6	Stead and Richards.
16	1100.0	88.7	10.3	0.0	Stead and Richards.

1. Soft Bessemer steel with few blow-holes; 2. Rail steel, many blow-holes; 3. Rail steel, no blow-holes; 4. Soft Bessemer steel, many blow-holes; 5. The same, few blow-holes; 6. The same, few blow-holes; 7. The same, very few blow-holes; 8. Bessemer rail-steel; 9. The same; 10. The same; 11. Bessemer spring steel, few blow-holes; 12. Open-hearth steel, before addition of spiegeleisen; 13. Bessemer steel, before adding spiegeleisen; 14. The same; 15. Dense steel, drilled with sharp drill; 16. The same, drilled with blunt drill.

These analyses show that liberated hydrogen and nitrogen, especially hydrogen, evolved by the solidifying steel, accumulate in larger or smaller bubbles, and form the blow-holes.

It may seem strange, on first thought, that molten steel should be able to absorb so much hydrogen, since this gas cannot be present in large quantities under ordinary conditions in the atmosphere of our steel-furnaces. We know, however, that the tetravalent metals of Mendelejeff's group VIII.—viz., iron, nickel, cobalt, ruthenium, rhodium, palladium, osmium, iridium, and platinum—have a characteristic high affinity for hydrogen, which will naturally be augmented when the gas is *in statu nascendi*, in which state it will combine much more readily with the metal. In our Bessemer and open-hearth proc-

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esses we decompose the moisture, brought in by the air and gas respectively, so that a considerable amount of nascent hydrogen is present during the process of manufacturing steel.

IV. In What Condition are Hydrogen and Nitrogen Present in Steel?

Many facts tend to show that we have in hydrogen not a metalloid, but the vapor of a metal which volatilizes very easily at extremely low temperature.

The high thermal conductivity of hydrogen resembles that of metals; the hydrogen is easily replaced in its combinations with metalloids partly or completely by metals; thus, $HCl + Na = NaCl + H$. Graham took hydrogen, therefore, as the vapor of a metal, which he called hydrogenium.[11]

Iron, like platinum, palladium, cobalt, and nickel, readily absorbs hydrogen in volumes many times larger than its own, especially when hydrogen, as we have seen by electrolyzing a solution of ferric chloride, is present *in statu nascendi*. This iron, obtained by electrolytic precipitation, is extremely brittle and very hard. Were we to assume only a mechanical absorption of the gas by the iron, a decrease in the hardness of the metal should be expected. When we regard the hydrogen as alloyed with the iron, however, the increase in hardness is quite natural, since all true alloys generally exhibit a greater hardness than the single constituent metals. If the hydrogen is alloyed with the iron, then it must be subjected to segregation, just as other alloys are. In what form the hydrogen is present in the iron is yet unknown; so much, however, seems to be sure, that that part of the hydrogen, present in the iron, which

[11] The phenomena of gas-absorption which some metals show, though destitute of visible pores, is probably connected with their capacity to absorb gases either on the surface or in their masses. Graham found that platinum occluded four times, and iron 0.44 times, its volume of hydrogen. This property is most remarkable in palladium, which absorbs hydrogen not only in cooling after being heated, but also in the cold. Palladium, used as electrode in the decomposition of water, will absorb 980 times its volume of hydrogen. This gas is again driven out on being heated. By the occlusion of hydrogen the volume of palladium is increased by 0.09827 of its original amount, from which it follows, that the hydrogen which under ordinary circumstances has a density of 0.000089546 that of water, has here a density nearly 9,868 times as great, or about 0.88 that of water. Hence, the hydrogen must be in the liquid or even solid (metallic) state. It probably forms thus an alloy with palladium like a true metal.—Ganot, *Éléments de Physique*.

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exerts such a great influence on the physical properties of the metal, will not consist of merely mechanically occluded gas, but must be alloyed with the metal.

The investigations in regard to nitrogen in iron are very scant so far. Older investigations by Fremy, Rinmann and Allen show that steel contains not more than 0.008 per cent. of nitrogen when manufactured from the liquid state and allowed no chance by further working to take up nitrogen. H. Braune has lately proved that solid steel, which is able to take up nitrogen, when heated up to from 600° to 800° C. in dry ammonia gas, is remarkably influenced in regard to its physical properties by the nitrogen thus absorbed. Nitrogen, like hydrogen, must therefore have the capacity to combine with iron.

Whether these two gases, when taken up by the molten steel, enter completely into combination with the metal or are only partly mechanically absorbed by the steel, cannot be decided, since we are unable to analyze molten steel.

V. Are Hydrogen and Nitrogen Alone Responsible for the Formation of Blow-Holes?

The analyses of the blow-hole gases show that practically only hydrogen and nitrogen are present, and we are, therefore, apparently justified in concluding that blow-holes are formed only because these two gases were absorbed by the molten steel in quantities too large to be kept in bond completely by the solidifying metal. The gases evolved by the slowly freezing steel should, therefore, be hydrogen and nitrogen only. But in case we find some other gas evolved from the ingot in appreciable quantity besides these, we are warranted in inquiring whether this third gas has not more to do with the formation of blow-holes than the analyses of the blow-hole gases would lead us to believe.

Dr. F. C. Müller,[12] by analyzing the gases evolved from the slowly solidifying ingot, obtained the results shown in Table II.

[12] *Stahl und Eisen*, vol. iii., p. 443 (1883).

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TABLE II.—*Analysis of Gases from Cooling Ingots*, by Dr. F. C. Müller.

No.	CO. Per Cent.	H_2. Per Cent.	N_2. Per Cent.	CO_2. Per Cent.	Remarks.	
1	37.3	47.3	7.9	7.5	Bessemer rail steel.	Finished steel, after addition of deoxidizing material.
2	34.0	49.5	8.6	7.9	Bessemer spring steel.	
3	43.2	19.7	36.7	0.4	Bessemer spring steel.	
4	45.9	41.4	9.9	2.8	Bessemer spring steel.	
5	38.7	51.3	7.2	2.8	Bessemer rail steel.	
6	48.2	44.5	2.5	4.8	Bessemer spring steel.	
7	38.4	51.0	2.2	8.4	Bessemer rail steel.	
8	31.5	39.2	21.6	7.7	Bessemer rail steel.	
9	68.0	16.2	11.0	4.8	Thomas rail steel.	Before addition of deoxidizer.
10	56.5	33.0	8.6	1.9	Thomas rail steel.	
11	18.6	54.2	24.9	2.3	Bessemer blown metal.	
12	18.1	52.9	26.8	2.2	Bessemer blown metal.	

We see that carbon monoxide is present in very much larger percentage in the gases evolved by the solidifying steel than in the blow-holes in the solid ingot. During some time, at least, carbon monoxide must, therefore, have something to do with the liberation of the gases occluded in the steel. What *rôle* it plays we cannot tell yet; but that it is concerned in the liberation of the gases is obvious. Another observation from our daily practice makes it still more likely that we have in the carbon monoxide an agent more or less directly responsible for the evolution of the occluded gases, and consequently for the formation of blow-holes. When we blow a Bessemer heat, and stop the blow just at the proper time, the steel will pour quietly after we have added the deoxidizer, and will give ingots free from blow-holes. But when we blow another heat, being very careful to have practically the same temperature as with the first, but continue the blowing for a short overtime (about 30 to 60 sec.); in other words, when we overblow the heat, the steel will pour decidedly wilder after the addition of the deoxidizer; it will rise in the molds and will give ingots with more or less broad rings of blow-holes.

Have the 30 to 60 sec. of longer blowing with the second heat overloaded the steel with hydrogen and nitrogen to such an extent as to account for the formation of the blow-holes? Is it possible that this short overblow is so much more powerful to saturate the steel with hydrogen and nitrogen than the much longer period preceding the overblow, during which the

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temperature of the bath was practically at the very same height? Or may it not be due to the fact, that during this very short overblow we form a metallic oxide in the bath, which was less liable to be stable as long as the more easily oxidizable impurities were present in still appreciable amount? And could not this oxide, reacting with the carbon brought in by the deoxidizer, give rise to a considerable carbon monoxide formation, which latter gas then had a decided influence upon the liberation of hydrogen and nitrogen in the steel?

V. Source of the Carbon Monoxide Gas.

In the manufacture of Bessemer or open-hearth steel, we take advantage of the great affinity for oxygen of the impurities contained in the iron. In the converter, we oxidize them by blowing atmospheric air through the molten metal, while in the open hearth we derive the necessary oxygen from the air and from iron oxides (iron-ore).

This oxidation calls for large quantities of oxygen; we always add a surplus of this element to our bath, or else the required reactions would not be completed in so short a time as is the case.

A part of this supplied oxygen will, therefore, stay combined with iron, which, as we know, has a great affinity for oxygen itself. In the presence of such a large surplus of metallic iron as we have in these steel processes, ferrous oxide, the most stable of the iron oxides, will be formed.

A. Ledebur found as much as 1.30 per cent. of ferrous oxide in low-carbon steel. This he found in solid steel; how much molten steel is able to contain it is impossible to say. It must be more than in solid steel, for otherwise it would be impossible for low-carbon steel (see Table II., analyses Nos. 11 and 12) to give off such large quantities of carbon monoxide.

The solvent power of steel for ferrous oxide increases with increase in purity. As long as the metal is still high in impurities, the ferrous oxide will become reduced by the more easily oxidizable elements, silicon, manganese, phosphorus and carbon. This reduction of ferrous oxide will be aided by the lower temperature, which naturally prevails in such a bath. As the temperature rises, however, the solvent power of steel for ferrous oxide increases.

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We know that it is much harder to eliminate phosphorus in the open-hearth furnace at high than at low temperature, although the oxygen at hand to combine with phosphorus is by no means smaller in the former case than in the latter. But the fact that under such circumstances, even carbon, which may often be present in appreciable percentage, does not readily reduce the ferrous oxide, is somewhat bewildering, since we know that the affinity of carbon for oxygen increases with rising temperature. We cannot manufacture very low carbon steel in the open-hearth furnace or converter, without greatly overoxidizing the metal—that is, increasing the amount of ferrous oxide in it; and we are forced to conclude that at high temperature the affinity of iron for oxygen must be greater than that of carbon for oxygen. Whether in this case the iron combines with oxygen to form ferrous oxide or, perhaps, to form a still lower oxide (F_2O), it is yet impossible to decide.

Our daily practice teaches the same law. If, from a heat high in carbon, but low in temperature, we pour a small sample with a spoon into a test-mold, the steel will solidify quietly in the mold. It will not sputter, and will not show any blow-holes when broken in two. But when we take a similar test from a heat of practically the same carbon-content, and much higher temperature, the steel will be "wild," will sputter and rise considerably; and when broken the ingot will show many blow-holes. What is the reason for this wide difference of behavior? In the first case, although ferrous oxide was certainly formed continuously in the bath, the metal refused to dissolve it, and the carbon, having a greater affinity for oxygen than iron, immediately reduced it. Hence, the metal poured into the test-mold did not contain ferrous oxide, and, the formation of carbon monoxide in the solidifying steel being consequently impossible, the steel solidified quietly without forming blow-holes. In the second case, however, the much higher temperature of the bath increased the solvent power of the steel for ferrous oxide, while, on the other hand, the affinity of carbon for oxygen was becoming less than that of iron for oxygen. Hence, the metal poured into the test-mold contained, in the presence of carbon, an appreciable amount of ferrous oxide. The rapidly sinking temperature of the steel, approaching its freezing-point, lowered simultaneously the solvent

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power of the steel for ferrous oxide and raised the affinity of the carbon for oxygen, so that a large amount of carbon monoxide was set free, which liberated hydrogen and nitrogen from the metal, and which were entangled in the metal as it passed through the pasty condition; and thus the blow-holes were formed.

In a low-carbon heat the conditions for the solution of ferrous oxide are much more favorable; since both high temperature and the higher degree of purity tend to increase the solvent power of the bath in that respect.

But what happens when we cool down such a heat artificially in the furnace? Prof. Howe says: [13]

"Certain metallurgists . . . point to their observation [in order to prove their statement, that the solubility of gases in molten steel decreases as the temperature falls towards the freezing-point], that when in the open-hearth furnace the charge has ceased to boil, boiling may be induced by shutting off the supply of gas altogether, which no doubt lowers the temperature."

I must confess here, that I have endeavored many times, without success, to verify this statement. I have taken heats with carbon close to 0.40 per cent., and with carbon close to 0.15 per cent., which had ceased to boil, and shut off the gas for 10, 15, 20 minutes and more, but never could detect a renewed boiling in the bath. All that I could observe was, that the slag cooled off rapidly on the surface, and therefore quickly became foamy under the bloating influence of the escaping gases (mostly carbon monoxide), which had been discharged quietly through the slag as long as the gas-flame kept it fluid and therefore easy to be penetrated. Moreover, I cannot believe that any considerable amount of cooling can take place in the bath lying, as it does, on the extremely hot hearth of the furnace, and protected by a more or less deep layer of slag. What had been believed to be a renewal of the boiling was nothing more than a change in the physical condition of the slag surface.

To cool a heat in the open-hearth furnace artificially, we may throw in pieces, not of pig-iron, which, for obvious reasons, will induce a boiling, but of steel scrap, which, in melting, cools the surrounding steel, and causes a violent local boiling.

[13] In his paper, already cited, p. 72, footnote.

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Or, still better, we may stir the bath with a steel bar. A high wave or crest will form immediately over the bar. The slag, unable to release readily the much larger quantity of gas, will foam up, sometimes so much as to run out of the furnace-doors. That the gas thus escaping is chiefly carbon monoxide, from the reduction of ferrous oxide by carbon, is proved by the fact that the stirring, as soon as a strong liberation of gas takes place, lowers the carbon-content in the bath considerably. This phenomenon, however, might conceivably be due to the fact, that in a steel bath, ferrous oxide and especially carbon, when present in small quantities only, will not react very readily upon each other. Both may possibly be in a state of inertia, which may be overcome by stirring or throwing in cold scrap. But opposed to this supposition, it may be stated that the liberation of gas ceases practically at once, when the steel scrap has melted, or the stirring is discontinued.

Moreover, the evolution of gas can be seen only at that locality where the scrap or the bar is at the very moment, and stops at once as soon as the scrap or the bar is moved away to some other part of the bath. The wave is always directly over the bar; if we move the bar quickly, the wave quite naturally lags behind somewhat, but still follows the bar, and subsides rapidly when the bar is removed. Were it only a state of inertia that had to be overcome, why should the reaction not continue and spread over all the bath after we had overcome the inertia in one part of the bath? We are apparently justified in concluding that the cold scrap or the stirring-rod effected a drop in the temperature of the steel in its immediate neighborhood only, and that this drop in temperature enabled the carbon to reduce the ferrous oxide, for which the solvent power of the steel was lessened simultaneously. To sum up:

(*d*) Although we find practically only hydrogen and nitrogen inclosed in blow-holes, these two gases are not alone responsible for the formation of the blow-holes.

(*e*) An important factor in the formation of blow-holes is carbon monoxide, which is present in large quantities in the gases evolved from the solidifying steel.

(*f*) The source of this carbon monoxide is the ferrous oxide, dissolved by the steel during the process of steel-making.

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(*g*) Steel has a solvent power for ferrous oxide, which increases with rising temperature.

(*h*) Steel has a solvent power for ferrous oxide, which increases with increasing degree of purity.

(*i*) The affinity of iron for oxygen at higher temperature increases with rising temperature more rapidly than that of carbon for oxygen.

VII. The Formation of the Blow-Holes.

Let us consider a metal which is very liable to form blow-holes, a metal, therefore, which not only contains much hydrogen and nitrogen, but also ferrous oxide—in other words, an overblown or over-ored metal. When this metal is tapped into the ladle we observe, on adding the deoxidizing or recarbonizing materials, a strong reaction. These additions are always, even when they consist of molten spiegel or ferromanganese, decidedly colder than the molten steel. A sharp drop in temperature must take place, and ferrous oxide, reduced by carbon, forms carbon monoxide, which is not absorbed by the steel, but leaves the metal immediately on its formation. Rushing out of the liquid metal, this gas will act on that hydrogen and nitrogen, which is present in the molten steel as mechanically absorbed gas, just like air or some other gas will act on carbon dioxide, when blown through mineral water in a tumbler. Here a physical law comes into effect—viz., that when one gas is set free in a solution containing several gases, this liberated gas will set free also the other gases. Besides carbon monoxide we find, therefore, also hydrogen and nitrogen leaving the steel. J. E. Stead, analyzing the gases accumulated under a slag cover in a ladle filled with steel, found: CO, 47.3; H_2, 18.6; N_2, 31.4; CO_2, 2.5 per cent.

Closely watching the ladle which now contains all the steel, under a more or less heavy cover of slag, we see that the evolution of gases subsides more and more, and that the slag on top begins to chill rapidly, with the exception of a small ring close to the ladle-lining. Here the evolution of gases continues, and, bringing constantly fresh, hot slag to the surface, keeps the slag liquid and in constant motion. Steam, coming from a still moist ladle-lining, cannot be the cause of it, since the ladle was not only perfectly dry, but also heated

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up to redness before the heat was tapped into it. The steel nearest the lining cools down somewhat, effecting the formation of carbon monoxide, hence an evolution of occluded gases from this locality. The cooling-effect of the ladle-lining is not great, but it lasts for a long time.

When we pour this metal into the molds we can often see very clearly that, after the mold is filled, the steel stays perfectly quiet in the mold for some time, and then begins to evolve gases and to rise quite suddenly. This phenomenon may be seen more readily in molds of smaller diameter than in larger ones.

What has happened in the meantime in the ingot? As soon as the steel enters the mold it experiences a sharp drop in temperature, where it comes in contact with the comparatively cold walls. Here it solidifies immediately, forming a shell, which incloses the liquid steel. This first shell contracts and draws away from the mold, leaving between ingot and mold a small space. This will certainly retard the freezing of the still liquid steel, which will now begin to solidify in layers, concentric with this shell and in perfect union with it. From the freezing of water and alloys we know that in the endeavor to crystallize out in pure iron crystals the steel will expel the impurities, which, having a lower melting-point than the purer metal, will travel towards the still fluid center of the ingot. We find, therefore, that the layers first solidified contain less impurities than the average of the original steel mass; and thus we have a purer, already solidified shell on the outside, and a liquid central part, where the impurities are still present in original amount, while between these two portions is a layer, which contains not alone the original amount of impurities, but also those rejected by the solidifying outer layer. This intermediate layer, which is still liquid, has not only taken up the carbon, manganese, phosphorus, sulphur, etc., expelled by the freezing outer layer, but also the hydrogen, nitrogen, and ferrous oxide, which were driven out of the solidifying crust.

These impurities take time to migrate through the fluid steel, and there will consequently be a gradually rising wave of impurities, traveling towards the center of the steel ingot—and also, on account of their lighter specific weight, upwards

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in the ingot—leaving behind it metal which is purer than it was originally, and facing metal still of original composition.

This wave, containing the highest amount of impurities per unit of steel, will naturally have a lower melting-point than the original steel mass. The cooling-effect of the crust will work on its outer side, and the heating-effect of the fluid central part on its inner side. If the impurities in this intermediate zone comprise enough carbon and ferrous oxide to permit their mutual reaction, we shall have: $FeO + C = Fe + CO$.

This reaction naturally must start where the intermediate zone is the coolest—*i. e.*, on its outer side, next to the first solidified layer—because, the solvent power of steel for ferrous oxide increases and decreases with its purity, and also with its temperature; and this zone is one of decreasing purity and falling temperature. Here, therefore, the carbon affinity overpowers the iron affinity for oxygen, and the reaction commences, liberating carbon monoxide gas, which, unable to become absorbed by the metal, tries to escape through the still liquid steel. In its passage, it forces the mechanically absorbed hydrogen and nitrogen also out of the steel, causing a strong evolution of gas in the ingot.

We must, however, discuss the above reaction a little further. We know that the reduction of ferrous oxide by carbon is not performed without absorption of heat. Taking the atomic weights, we have: 72 ferrous oxide + 12 carbon = 56 iron + 28 carbon monoxide. That is to say, 1 lb. of carbon, oxidized to carbon monoxide, reduces 4.7 lb. of iron from ferrous oxide.

This oxidation of 1 lb. of carbon to CO, evolves 2,470 calories, while the reduction of 4.7 lb. of iron from FeO requires $4.7 \times 1{,}350 = 6{,}345$ calories. We have, therefore, a loss of 3,875 calories for every pound of carbon thus oxidized.

The reaction which takes place in the intermediate ring—first in its outer part—between carbon and ferrous oxide, is, therefore, a strongly endothermic one, and must have a decided cooling-effect on the surrounding steel, from which it draws the heat necessary for its completion. Hence, the drop in temperature must be sharp in the intermediate ring, with the natural further consequence that the reaction between carbon and ferrous oxide must proceed very rapidly through that ring. The steel in the outer part of this intermediate ring must sud-

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denly enter the pasty state, which condition will quickly be acquired by those parts facing the central part of the ingot, since it follows closely the rapid progress of dropping temperature through the ring.

Some of the evolved gases will find their way out of the steel; the rest will become entangled in the network of the pasty metal, and form blow-holes. This process will be repeated as long as the successive layers are rich enough in ferrous oxide and carbon to insure the carbon monoxide formation.

By the appearance of the blow-holes in the solid ingot we can judge how thick each intermediate ring was when the carbon monoxide formation took place; for whenever the reaction in one individual ring was over the blow-holes formed tried to close in. The blow-hole tube will therefore experience a strangulation, until the reaction, taking place in the adjacent ring, opens the blow-hole tube again to its former size—or even makes it wider than it was originally, on account of the slower passage of this next ring through the pasty state, which gives the gases more opportunity to accumulate in one place.

This cooling-effect of the reaction $FeO + C = CO + Fe$ and its natural consequence on the formation of blow-holes, has, as far as I know, never been discussed before. I consider it of the greatest importance in the formation of blow-holes, since I believe that, all other conditions being equal, that steel will in every case form the largest amount of blow-holes which contained the greatest amount of ferrous oxide, or which, in other words, was most overoxidized.

The above-described reaction destroys the largest part of the ferrous oxide present in the intermediate ring. What is left cannot leave the solidifying steel as easily as the other impurities, since it cannot possibly coalesce as easily as the latter, and therefore cannot segregate out of the freezing metal. Being itself very refractory, it will remain finely disseminated in the solidifying steel. Ferrous oxide segregates freely in liquid steel only, not in steel approaching the freezing-point.

After the reaction between ferrous oxide and carbon is over in all those more or less wide intermediate rings which contained enough to insure it, and after blow-holes have been formed in those intermediate rings, forming together one more or less broad ring of blow-holes, another stage—namely, the

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uninterrupted cooling and freezing of the central but still fluid part—can follow.

The impurities contained in the solidifying layers, and expelled by them on freezing, will again accumulate in a ring, until the reaction between ferrous oxide and carbon again becomes possible, whereupon a second ring of blow-holes will be formed in the ingot. This second ring will, however, consist of smaller individual blow-holes than the first; since the liberated gases, traveling upwards through the liquid steel, have lessened throughout its contents of occluded gases, so that the amount of gases available to form blow-holes in the second ring must be decidedly smaller than in the first. This effect proceeding from the first ring may, indeed, be so strong as to reduce the gas-content of the remaining steel until it can solidify without further blow-hole formation. In that case, the ingot will have only one ring of blow-holes.

It must be remembered, also, that the migration of the ferrous oxide, like that of the other impurities, is not only horizontal but vertical in the fluid steel, and that the steel which it left behind becomes purer and purer. However, the richer the fluid steel becomes in impurities, the lower its melting-point will drop, with the consequence, that the liberated carbon monoxide together with the other gases has continuously increasing opportunity to leave the metal.

The hotter we pour steel, the more ferrous oxide we must reckon with as contained in it to start with. The overloading of the intermediate ring with ferrous oxide will therefore occur earlier, the reaction between ferrous oxide and carbon will take place quicker, and the blow-holes formed will lie closer to the ingot-surface.

Some broken ingots which I have examined showed blow-holes without any "strangulation" rings, but with practically perfect, smooth walls. These blow-holes, which were of considerable size, although never as long as the long blow-hole tubes with "strangulation" rings, had apparently been made without any interruption. They must have been cast in one mold, as it were! When the intermediate ring has been formed, and is liable at any time to bring about the reaction between ferrous oxide and carbon, it is indeed possible that, under favorable conditions, undercooling may take place in this

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ring—*i. e.*, the temperature throughout this ring may drop below that point, where under less favorable conditions carbon monoxide would have been made under all circumstances. If this undercooled ring now experiences only the slightest change in its conditions, the carbon monoxide reaction will not proceed from the outer part gradually towards the inner part, but the reaction between ferrous oxide and carbon will set in suddenly throughout the entire ring, so that the blow-holes will be formed all at once. These blow-holes will show smooth walls as the consequence of their uninterrupted formation.

The practical steel-maker knows that high-carbon as well as low-carbon steel will pour quietly, and that the ingots will be free from any dangerous blow-holes, if he is able either to prevent or to destroy overoxidation in the bath before teeming the steel. He knows also that he can bring about this state of affairs by keeping the temperature during the process well under control and getting the slag into the right condition; and he is aware that heats, which have been worked and finished too hot, will not pour quietly, and will give "spongy" steel.

The more completely we control the temperature, therefore, the more we shall be able to get steel into the molds, which solidifies solidly for a considerable period, before blow-holes are formed. The steel will have a solid crust of substantial thickness; its blow-holes will be deep-seated and hence less harmful.

Steel at a very high temperature, and containing an undue amount of ferrous oxide, is pretty hard to deoxidize successfully. If, however, the overoxidation is destroyed, such steel, although poured extremely hot, will give ingots free from blow-holes.

When steel at too low a temperature is poured into molds without being sufficiently deoxidized, blow-holes will be formed scattered all through the ingot. Here a regular ring-like solidification of the steel could not take place, as with steel of higher temperature. The segregation of the impurities is irregular—a little here, much more there—and the consequence is, that the reaction of ferrous oxide and carbon will be confined to smaller, irregularly distributed areas. That the result shows masses of blow-holes scattered without rule through the entire mass is not surprising.

Another phenomenon which I wish to mention in connection with the formation of blow-holes is the well-known appearance,

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in freshly stripped ingots, of a darker lower, and a brighter upper, part. When we cut such an ingot in two, we find that in the darker lower part the blow-holes are close to the surface, while in the brighter upper part the blow-holes are much farther away. This difference is easily explained upon the foregoing theory. When the lower part of the steel begins to solidify and consequently to expel gases, the upper part will be still fluid; first, because the steel there has not been in the mold so long; and second, it is resting on the hot steel of the lower part, like a pot on an open fire, and not, like the steel in the lower part, on the heat-absorbing cold stool. The steel in the upper part has the gases partly washed out by the passage of the gases, expelled by the steel underneath. The upper part, therefore, will be much poorer in gas when it begins to solidify; the period for a quiet and undisturbed freezing will be longer; and the first blow-hole ring will lie deeper in the ingot. The closer the blow-holes are to the surface, the less metal will there be, through which the heat from the still fluid ingot-center is sent to the outside, and less heat will be conducted to the surface—hence the darker appearance of the lower ingot part. The thicker the solid ingot-crust the more heat will be conducted through the solid metal to the surface; and the more will this surface be heated—hence the brighter appearance of the upper ingot part.

The fact that the upper part of the still fluid metal inside the ingot is hotter than that in the lower part, only strengthens this difference in the appearance of the two parts.

On page 71 in his paper Professor Howe says[14] that under certain conditions

". . . . the blow-holes are so large as to be harmful, and they cannot be effaced by welding, because they lie so near the skin of the ingot that their walls are oxidized by the infiltering atmospheric oxygen, so that the contact of metal with metal, necessary to welding, is lacking. But (under other conditions) . . . . the blow-holes which form are so deep-seated as to be harmless, because their sides will not be oxidized. and therefore they will weld up completely in rolling, and will thus disappear."

With this statement that, under favorable conditions, blow-holes will weld up completely in rolling, both theoretical reasoning and practical experience render me unable to agree.

[14] This volume, p. 71.

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The gases, which, as we have seen, fill the blow-holes in the steel ingot, stay in them from the very first moment these holes are made, until the steel is rolled, finished and ready for shipment. They are not reabsorbed by the steel; this would be impossible. Nor are they driven out of the deep-seated blow-holes. They are found even in steel which has been worked very extensively by forging down to small bars.

Drilling such forged steel, Dr. Müller found:

Volume of Gas.	Composition of Gas.		
	H_2.	N_2.	CO.
Per Cent.	Per Cent.	Per Cent.	Per Cent.
5.0	52.2	48.1	0.00
7.3	54.9	45.5	0.00
5.5	72.4	25.3	1.30

The composition of this gas is conspicuously similar to that of the original blow-hole gas. That the amount of gas obtained is so much less with forged steel than with unworked is quite natural, since the reduction in size by forging brings the solid and the cavity-bearing parts of the original steel block close together, so that the unit of drilled steel contains much less gas in forged than in unworked steel. Through the entire period of rolling, forging, etc., the blow-holes are filled with gas, which, being under a pressure of at least three atmospheres in the cold steel, is naturally under a much higher pressure while the steel is at the rolling-temperature. The action of rolling and stretching the steel will also stretch the blow-holes, without interfering with the contained gas, which will remain in them just as air remains in a partly-filled rubber bag, which is passed to and fro between two rollers, some distance apart.

Now, welding requires, as Prof. Howe says, the contact of metal with metal, and he points out that the oxidation of the walls of blow-holes would prevent this metallic contact. But it may be prevented by other substances as well. The gas in a blow-hole, although it may be pressed from one end to the other, as the hole becomes during rolling longer and thinner, will never permit a complete welding of the metallic walls. Small portions of them may weld together; but any weld of larger extention would certainly be severed again by the gas, which in the reduced blow-hole would naturally be under a

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still higher pressure than originally, and would consequently be much more powerful to tear apart a fresh weld of the sides. The gas acts like a cushion always present, which will not permit any extended welding in even the deepest-seated blow-holes.

But besides this theoretical reason, my experience forbids me to believe in the welding of deep-seated blow-holes. The etched surfaces of many hundreds of rail sections have proved to me conclusively, that the original blow-holes are still present in the finished rail, though, of course, much smaller in diameter by reason of the rolling. The welding of blow-holes by rolling, forging, or any other operation, must therefore, in my judgment, be regarded as impossible.

VIII. The Prevention of Blow-Holes.

Having traced the causes of blow-holes, we may proceed to discuss the means by which they may be either prevented altogether, or, at least, located so deep in the ingot as to be less harmful, by reason of the presence of a surrounding mass of solid steel, thick enough to withstand the rolling without being torn apart.

Mr. Brinell's experiments and suggestions as to the proportions of manganese, silicon or aluminum in steel required to prevent blow-holes, form only a portion of the story, and will not help us very much, since by far the largest part of the daily product of Bessemer and open-hearth steel contains just that amount (in the neighborhood of 0.50 per cent.) of manganese plus silicon which he pronounces the most dangerous in this respect. As to the addition of aluminum, I must say that I have had many heats, which rolled very badly and showed all the signs of outside blow-holes, although aluminum had been added with the greatest care and discretion. On the other hand, many heats, containing the dangerous percentage in manganese and silicon, with no aluminum at all, rolled perfectly. If blow-holes could be prevented by adding a certain percentage of aluminum, the art of making sound steel ingots would lose all its difficulties—and perhaps also its interesting features!

It has been shown above, that the formation of blow-holes is not only due to the presence of hydrogen and nitrogen, but also—and not least—to the presence of ferrous oxide in the

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molten metal. Indeed, we must ascribe to the presence of ferrous oxide more influence than to any other condition, since without it the liberation of hydrogen and nitrogen would always be less explosive. The entanglement of the evolved gases in the steel, when it is becoming suddenly pasty, is greatly aided by the reaction of ferrous oxide and carbon, rapidly forming large quantities of carbon monoxide, and simultaneously lowering the temperature of the steel. We know that overblown heats give porous ingots, while heats blown for a shorter time (though often only a little shorter), and presenting no more overoxidation than is effectively destroyed by the deoxidizing additions, will pour quietly and give solid ingots. Overoxidation of the steel must therefore be considered to be the chief cause of blow-holes. To prevent this defect, we must endeavor either to prevent oxidation or to destroy the overoxidation of the molten steel before it enters the molds. If we are successful in this, we may rest assured that the ingots will have only deep-seated, harmless blow-holes, if any, and that their faultless rolling into rails, plates, billets and structural material, etc., is assured.

1. *The Prevention of Overoxidation.*

In the Bessemer process overoxidation is prevented by guarding against blowing too hot or too long. Both will overload the steel with ferrous oxide, which cannot be completely reduced by manganese, etc., in the ladle, but will enter the molds and continue therein the reaction with carbon and the dangerous formation of carbon monoxide.

In the open-hearth process, iron-ore is usually added to assist the oxidizing influence of the flame. Iron oxides are thus brought into the bath, giving the metal every opportunity to saturate itself with ferrous oxide. To prevent too high a degree of overoxidation at the end of the heat in the open-hearth care must be taken to control the temperature of the bath and the oxidizing influence of the slag. In this process, the slag is the transmitter of the oxygen, and also the receptacle of the oxidized impurities, except the oxides of carbon. If we finish a heat with a slag too rich in oxygen, we are liable to the very last moment before tapping to the transfer of ferrous oxide from the slag into the bath. If we have, at the same time, too high

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a temperature, we must expect a greater solvent power of the metal for ferrous oxide, irrespective of the carbon-content. The slag will give up ferrous oxide to the steel, which absorbs it only too readily. The bath of molten steel has a certain depth. All other conditions being equal, it will have the lowest temperature, and consequently the smallest solvent power for ferrous oxide, in its lowest layer, where it is in contact with the heat-absorbing lining of the hearth. The upper layers, close to the slag, will be the hottest, and will possess the greatest solvent power, which will be satisfied, not only by the ferrous oxide derived from the slag close at hand, but also by the ferrous oxide, which, being specifically lighter, will rise to these upper layers from the bath below. This may be easily proved, by taking two tests from the bath, one as close as possible to the bottom, and the other as close as possible to the slag. In the test-mold the steel from the lower part of the bath will solidify much more quietly, and will exhibit blow-holes to a much smaller degree, than that from the upper part.

The attainment of high temperature by the upper layers is assisted by a thick and heavy slag, which keeps the temperature back in the bath, whereas a fluid, thin slag transmits it more rapidly to the furnace and thence, with the waste gases, to the stack. It is impossible to get the bath hot enough to tap, when a very thin slag has been kept on it from the beginning. Hence, the slag must not be thinned until the bath has acquired a temperature high enough for successful tapping and pouring, and particular care must be taken to bring the slag to a high degree of fluidity. Such a slag will not allow the upper layers of the bath to take up an undue amount of ferrous oxide, since it keeps these layers from rising in temperature too far above the rest of the bath, and thus limits their solvent power for ferrous oxide.

In order to have, towards the end of the process, a slag not excessively oxidizing, care must be taken that the heat be not "over-ored," and that the last portion of ore added be perfectly "boiled out" before tapping. I have had opportunity to make many open-hearth heats with Swedish magnetite in large lumps. Proper care being taken in working down and finishing the heats, no trouble whatever was experienced with the resulting ingots. At other times, I had to use so-called "con-

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centrate" from Sweden—a fine, sand-like magnetite, the product of magnetic separation. In operating with this fine ore, besides taking much more time to bring down the carbon of the bath to the point required, I got extremely bad steel, full of blow-holes. This indicated that slag still rich in oxygen when the heat is ready to tap (as was the case with the fine magnetite concentrate) tends to the formation of overoxidized steel, and hence of blow-holes.

To prevent overoxidation in the open-hearth, care must therefore be taken:

(*k*) That the temperature during the process be kept not too high.

(*l*) That the slag be maintained in a good, thin condition during the latter part of the process.

(*m*) That the oxidizing power of the slag be brought to a minimum towards the end of the process.

But, though all these prescriptions be piously fulfilled, it will nevertheless be impossible to manufacture steel without any overoxidation at all. The solvent power of molten steel for ferrous oxide is so great that some of the oxide will be dissolved in the metal, in spite of all possible precautions. We must, therefore, reckon with a certain, unavoidable degree of overoxidation in all steel, and the next problem is therefore:

2. *The Destruction of Overoxidation.*

We have seen that when a stirring-rod is moved to and fro in the bath, a strong evolution of carbon monoxide takes place. Since this is just what we desire, we certainly should continue stirring until the wave over the rod has more or less completely subsided. This stirring of the bath, shortly before tapping, is greatly beneficial in lessening overoxidation in the steel, and should be more general than it is in present practice. Any addition to the bath of substances which combine easily with oxygen will naturally assist in the destruction of overoxidation.

Spiegel, ferromanganese or pig-iron (cold or fluid) will thus add carbon and manganese, both of which will combine with oxygen all the more easily, since their solution and the consequent reaction with ferrous oxide are accompanied with a lowering in the temperature of the bath. While the carbon monox-

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ide thus formed is escaping as a gas, the manganese protoxide will be taken into the slag; and there it will stay, unless the slag be so rich in manganese as to permit its reduction and return to the bath. But such a continuous reduction of manganese from the slag would decidedly aid in lessening overoxidation in the bath; and therefore it would be advisable to secure, in the open-hearth furnace, slags rich in manganese, if manganese (in any form) were not so expensive.

As to the deoxidizing influence of the silicon brought in by ferrosilicon, it may be said that silicon by itself will destroy overoxidation, but that such deoxidized steel will not be free from red-shortness. Only when manganese is present, besides silicon, will the red-shortness of the steel disappear.

3. *Heat-Finishing in the Ladle Versus Heat-Finishing in the Furnace.*

It is the general custom in this country to deoxidize the steel in the ladle, while in many European works this practice is not regarded as safe, and the steel is deoxidized in the furnace before tapping. All other conditions being the same, deoxidation in the furnace is unquestionably better than in the ladle. The deoxidizer, added to the bath in the furnace, will float, partly submerged, on the metal; will dissolve there; and will therefore deoxidize just that portion of the bath which is most overoxidized. The only drawback is, that a part of the added manganese, etc., will be lost into the slag, which, by reason of its oxygen-content, is only too eager to seize upon the manganese, etc., and will thus deprive the bath, for which the addition was intended, of part of its deoxidizing supply. To counterbalance this loss, a certain surplus of the deoxidizer must be added, when deoxidation is performed in the furnace.

Deoxidation in the ladle, on the other hand, is generally satisfactory, unless the degree of overoxidation in the steel is unusually high. But in that case most of the deoxidizer will already be dissolved, before that portion of the bath gets into the ladle, which needs deoxidation the most—namely, the upper layers. The ferrous oxide of this portion, on reaching the ladle, will find the deoxidizer present, not in concentrated form and therefore most efficient, but already dissolved and strongly diluted in the large mass of molten steel. The

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reactions between carbon, manganese, silicon, and ferrous oxide are then necessarily more sluggish, require more time. It is possible that, in such cases, the steel may enter the molds, without having been properly deoxidized, and may consequently form dangerous blow-holes in the ingots. Another drawback to ladle-deoxidation is, that the manganese protoxide and silicate thus formed may often have too little time to separate into the slag before the steel reaches the mold and solidifies therein. The ingot may then contain small, but nevertheless dangerous, inclosures of slag (*i. e.*, manganese protoxide and silicate), which will make the steel "short" and brittle.

X. The Influence of Carbon, Manganese, Silicon and Aluminum in Steel Towards the Prevention of Blow-Holes.

Iron with at least 2.3 per cent. of carbon and various amounts of other impurities, such as manganese, silicon, phosphorus, sulphur, etc., is classed as pig-iron. It is distinguished chemically from steel (or ingot-iron) by the higher percentage of these impurities, especially of carbon. We have seen that iron in the molten state is able to absorb gases, and to expel them again on freezing. The amount of gases taken up by iron in the blast-furnace probably depends on the pressure in the hearth of the furnace, the degree of superheating in the hearth, the percentage of silicon in the metal, the melting-point of the iron and the slag, and the percentage of moisture in the blast.

The more gases the iron was able to absorb, while in the blast-furnace, the more it will try to set free, when solidifying, so far as its degree of fluidity will permit. The more fluid the metal, the more gases it will allow to escape. The thicker the iron, the more gases will stay in it and the greater the possibility of blow-holes. Iron made during a cold, irregular run of the blast-furnace generally flows very thickly, and exhibits many blow-holes. Tables III. and IV. illustrate these propositions.

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TABLE III.—*Composition of Gases Escaping from Molten Iron.*[15]

	H_2. Per Cent.	CO. Per Cent.	CO_2. Per Cent.	O_2. Per Cent.
(*a*) Foundry iron, thick and mushy, average from 13 analyses, ...	6.4	4.5	8.6	0.5
(*b*) Basic iron and spiegeleisen, thin and fluid, average from 48 analyses, .	21.9	19.1	1.3	0.4
(*c*) Foundry iron, very thin and fluid, average from 6 analyses, ...	47.3	13.7	1.3	0.5

TABLE IV.—*Composition of Gases Obtained by Drilling Solid Foundry Iron.*

Size of Bore-Hole. cc.	Gas Obtained Under Ordinary Pressure. cc.	Composition. H_2. Per Cent.	Composition. CO. Per Cent.
61	26.2	57.6	0.7
55	16.6	42.2	1.9
61	31.6	59.0	0.9
58	47.6	66.1	0.6
48	17.0	59.0	3.5
55	32.2	46.2	1.2
53	31.2	56.7	1.3
65	74.8	43.6	0.3
55	76.6	67.3	0.2

Why is it that pig-iron, although absorbing in the molten state large amounts of gases, has not, by far, so great a tendency to form blow-holes as steel or ingot-iron?

To answer this question correctly we must find out whether the same conditions as to the formation of carbon monoxide exist in pig-iron as in steel and ingot-iron. We know, now, that molten iron increases in solvent power for ferrous oxide—the source of carbon monoxide—with rising temperature and growing degree of purity, and that, on the other hand, the affinity of carbon for oxygen lags behind that of iron the higher the temperature rises. In pig-iron, the melting-point of which is considerably below that of steel (especially low-carbon steel), the solvent power for ferrous oxide must naturally be decidedly smaller than in steel. Hence, in approaching the freezing-point, pig-iron cannot form as much carbon monoxide as steel can. The solidification of pig-iron, therefore, is accompanied, not by a sudden, but only by a quiet, liberation of absorbed gases, which continues until the iron is solid.

That molten pig-iron is able to dissolve some ferrous oxide,

[15] E. Münker, *Stahl und Eisen*, vol. xxiv., p. 23 (1904).

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which in cooling reacts with carbon to form carbon monoxide, will be seen from the analyses given above. It may be asked, why this carbon monoxide, even though less abundant than in cooling steel, does not cause the formation of blow-hole rings in the solidifying iron—consisting perhaps of smaller individual blow-holes, but present nevertheless! The answer is found in the sudden passage of pig-iron from the molten to the solid state, without first becoming pasty. It is the pasty steel, as we have seen, which entangles the liberated gases in its rapidly growing network, and (since it is pasty and can therefore be pushed aside somewhat) allows the small bubbles of liberated gases to accumulate in large blow-holes. Pig-iron does not know such a pasty state; it does not, therefore, catch the liberated gases and prevent them from escaping, but stays fluid until the last moment, when it becomes solid. The liberated gases can therefore escape during the entire period of approach to the freezing-point, and no intermediate pasty state holds them back in the metal before the freezing-point is reached. Those gases which are set free, in greater or smaller quantity, at the very instant of freezing can never give blow-holes, either in pig-iron or in high- or low-carbon steel. They will remain finely disseminated through the entire mass of metal.

The reasons why pig-iron does not form blow-holes on freezing are therefore:

(*n*) Its low solvent power for ferrous oxide;

(*o*) The absence of a pasty state, between the molten and the solid states.

We may now more easily understand the preventive influence of carbon, manganese, and silicon on the formation of blow-holes. All these elements, single or combined, lower the melting-point of the iron, which therefore becomes fluid at a lower temperature. As to the degree of overheating, carbon especially will not allow the metal to overstep a certain temperature in the furnace. It is, in the open-hearth, the true temperature-regulator, unerringly and immediately acting, when the temperature of the bath is unduly high, by increased oxidation and consequent lowering of the temperature. Carbon and the temperature of the bath must always be in equilibrium. The more carbon, therefore, the steel contains the lower will be its

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melting-point and its temperature, and consequently its solvent power for ferrous oxide. But the less ferrous oxide the steel contains, the more certain will be its successful destruction by deoxidizing additions, and the smaller the danger of blow-holes. Moreover, carbon, just like silicon and manganese, tends to shorten the pasty state of the solidifying steel, thus giving more time and opportunity for the escape of gases before solidification. For this reason it is extremely difficult to cast steels low in carbon, manganese, and silicon into ingots which do not contain broad rings of blow-holes. Steels high in carbon, manganese or silicon stay perfectly fluid much longer; the pasty state is considerably shortened, and the danger of blow-holes decidedly lessened.

By adding carbon, manganese, and silicon, in the furnace or in the ladle, we endeavor to destroy the ferrous oxide present in the metal. This destruction is assisted by the increasing affinity for oxygen which these elements acquire with falling temperature (while that of iron decreases simultaneously), and also by the lowering of the melting-point of the metal, which permits them to act with increasing intensity. To deoxidize steel of excessively high temperature is, therefore, extremely difficult; every steel-maker knows that. Only when the steel has the right medium temperature can he be sure of success; and the natural consequence is, that he avoids pouring steel too hot into the molds, knowing that it will still contain ferrous oxide, in spite of the deoxidizing additions previously made. The temperature of the steel when it enters the mold governs its later behavior therein. If it be too hot, and consequently still have ferrous oxide dissolved in it, the reactions between this compound and carbon, manganese, and silicon will continue while the metal is approaching its freezing-point in the mold, and the formation of blow-holes is inevitable. Only when we tap the heat at the right temperature, or when we wait until an excessively hot heat has cooled down in the ladle, so that we pour at a good, medium temperature, can we be successful in our endeavor to destroy overoxidation in the steel before it enters the mold.

Aluminum, like carbon, manganese, and silicon, shortens the pasty stage and lowers the melting-point of the cooling steel. Like them, also, it destroys ferrous oxide by reducing it to metallic iron. It is much stronger in this action than man-

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ganese. One part of manganese reduces 1.3; one part of aluminum reduces 4.0; and one part of silicon reduces 5.14 parts of ferrous oxide. Hence, aluminum, added to the steel with the necessary care and discretion, would be a much better deoxidizer than manganese. But to add aluminum in the mold, when it is practically filled to the top, has no influence on the metal underneath. The upper layers of steel in the mold, becoming perfectly deoxidized by the added aluminum, will instantaneously become quiet. No gases will escape vigorously from these upper layers, and no fresh and hot steel will consequently rise continuously to the surface, keeping the ingot there open. The quiet surface-metal, losing its heat rapidly, will quickly solidify and give a solid ingot-top. But underneath, in the still fluid steel, carbon monoxide gas will continue to form, and will liberate hydrogen and nitrogen in considerable quantities. Instead of escaping from the ingot, these gases are kept back in it, so that, in the slowly solidifying steel underneath, many blow-holes are formed. A continuous addition of small quantities of aluminum to the steel while pouring it into the mold has the decided disadvantage that the alumina thus formed has no time and no chance to separate out on the surface. The formation of carbon monoxide, and with it the separation of hydrogen and nitrogen, stops practically altogether, so that the alumina molecules find no assistance to rise to the surface of the ingot. They remain in the steel and make it short and brittle, which is practically as bad as to have dangerous surface blow-holes.

The only place to add aluminum is in the ladle. Here it can do its beneficial work in destroying the ferrous oxide in the steel, while the resulting alumina has time and opportunity to rise through the fluid steel to the surface and to become harmless.

The often heard explanation that silicon and aluminum render the steel more porous for occluded gases, and thus prevent the formation of blow-holes, seems to be not only far-fetched, but also incapable of proof. I consider it better to explain the action of these two elements from a purely chemical and thermal standpoint, as I have done above.

Aluminum absorbs gases (hydrogen) when in the molten state. J. W. Richards[16] found that molten aluminum will absorb

[16] *Aluminium: Its History, Occurrence, Properties, etc.*, 2d ed., p. 56 (1890).

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large quantities of gas, which is set free again when the metal solidifies. M. Dumas found in 200 g. of solid aluminum, occupying 80 cc., when heated *in vacuo*, 89.5 cc. of gas measured at 17° C. and 75 mm. pressure. The gas consisted of 1.5 cc. carbonic acid and 88 cc. hydrogen. Since aluminum takes up gases itself, when molten, it must favor the absorption of gas by steel, when present therein.

X. The Prevention of Blow-Holes by Increase and Decrease of Pressure.

Experience of our daily life teaches us that a fluid is able to absorb and keep in solution more gas when under pressure. A closed bottle filled with mineral water or champagne keeps the liquid under quite a pressure, due to the disengaged carbon dioxide in the bottle, which presses on the liquid. No sign of carbon dioxide can be observed in the liquid, even when we shake the bottle vigorously. As soon, however, as the bottle is opened the pressure is released, and a strong evolution of carbon dioxide all through the liquid takes place. Slowly the liberation of gas subsides, and after some time ceases altogether. If we now connect the bottle with a suction-pump and form a partial vacuum over the liquid, a vigorous renewed liberation of carbon dioxide will take place. In the same way, water which has ceased to boil will strongly boil again without further supply of heat when placed in a partial vacuum. According to Bunsen, for the same gas, the same liquid and the same temperature, the weight of gas absorbed is proportional to the pressure.

Molten iron behaves in this respect like a true liquid, and is therefore able to keep more gases in bond under higher, than under ordinary atmospheric, pressure. Hence all processes and arrangements which put solidifying steel under pressure tend to prevent the formation of blow-holes. In such a freezing steel the point of solidification is raised in accordance with the pressure; *i. e.*, the liquid steel becomes solid sooner under pressure than under ordinary circumstances. This more rapid passage from the liquid to the solid state must necessarily shorten the intermediate pasty state, and thus hinder the formation of blow-holes. In steel, cooling under pressure, the ferrous oxide will be only partly destroyed by carbon, the time

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of this reaction being shortened. Hence, less carbon monoxide is formed and a smaller quantity of other gases is forced out by the escaping carbon monoxide. Steel solidifying under high pressure will therefore exhibit but few blow-holes. The gases did not have time to accumulate and to form large bubbles, but remained finely disseminated through the entire mass of steel.

The "Whitworth," "Illingworth," "Harmet" and "Williams" processes, described in Professor H. M. Howe's paper, will therefore give steel comparatively free from blow-holes.

Molds which, like the so-called "bottle-top" molds, are so constructed that the rising steel must soon press against a rigidly closed top, will give, under favorable conditions, ingots with less blow-holes than "open-top" molds, which generally are not capped until the steel has solidified somewhat around the edges of the top. Arrangements to exert a pressure on the solidifying steel in the mold, by filling the space between ingot-top and mold-cap with steam or carbon dioxide and thus to get ingots free from blow-holes, have been tried extensively in this country as well as in Europe. So far as I am aware, however, they are nowhere used to-day.

The same effect as that of increased pressure in preventing blow-holes would be obtained if the pressure over the liquid or solidifying steel should be reduced. In that case, the gases would be able to escape in much larger quantity than under ordinary pressure; the remaining steel would experience a considerable diminution of its gas-content, and blow-holes would consequently be less liable to form. Moreover, the reduced pressure would lower the melting-point, keeping the metal fluid for a longer time, and would consequently assist the escape of gases.

XI. Other Arrangements and Propositions to Prevent Blow-Holes in Steel Ingots.

Starting from the idea that any liquid, containing occluded gases, will release them when stirred thoroughly, W. D. Allen, of Sheffield, England, employed an apparatus by which he stirred the steel in the ladle. The results were very encouraging; the steel thus stirred gave sound ingots, free from blow-holes. This idea, however, did not have any extended adop-

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tion by other steel-plants, owing very probably to the fact that such a stirring necessarily requires a very hot steel, or else a heavy skulling of the stirring-arms will quickly take place. F. Knaffl[17] proposed to shake the mold immediately after pouring, by raising a little the stool on which the mold rests and dropping it again on a solid support. Such a shaking of the steel must certainly be of assistance to the escaping gases; the metal will solidify with less blow-holes than if allowed to set undisturbed.

A number of schemes for pouring steel in a partial vacuum were patented years ago, but, so far as I know, never tried in practice. The evacuation of the converter, after the heat is blown, has been proposed by W. Durfee and N. B. Wittmann.

Sink-heads, kept fluid for some length of time, either by making the upper part of the mold less conductive of heat, or by heating the sink-head with coke, gas or electricity, will also allow more gases to escape from the steel and will therefore assist in the prevention of blow-holes. Such sink-heads are used for the prevention of pipes in steel ingots, and ingots thus treated will doubtless be also improved as to blow-holes.

The idea, introduced by Boulton, of pouring a continuous ingot by putting always a new "open-top" mold on the already partly filled mold, and thus getting ingots free from blow-holes, has been in practical use at the West Bergen steel-works. The steel in the lower molds had opportunity to release the largest amount of its liberated gases through the still fluid metal above, with the consequence, that in the solidifying steel but a small quantity of gas was left, which was unable to form blow-holes of appreciable size and number.

Centrifugal force for degasifying liquid steel has been, and still is, employed in the manufacture of car-wheels. The very great difference in the specific gravity between liquid steel and gas must force the latter out of the molten metal, when put under the influence of centrifugal force. No doubt the centrifugal treatment of liquid steel before it enters the molds would greatly assist in the manufacture of ingots free from blow-holes.

To sum up: the means for the prevention of blow-holes in steel ingots are:

[17] German patent No. 48,874, and British patent No. 5,639 (1889).

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1. Medium temperature of the heat during the last period of the process in the converter or open-hearth.

2. Careful avoidance of overblowing or over-oreing of the heat; careful boiling-out of the last portion of ore added to the bath.

3. A finishing slag, not too rich in oxygen and having the proper degree of fluidity.

4. The destruction, by stirring the heat before tapping, of the ferrous oxide formed.

5. Addition of sufficient deoxidizing material to the heat, and the allowance of sufficient time for the complete separation of the manganese protoxide, silicate of manganese or alumina, thus formed, into the slag.

XII. The Damage Done by Blow-Holes in Steel Ingots.

We must distinguish between deep-seated blow-holes and those which lie beneath the ingot-skin. While the former are practically harmless, since they are not affected by the oxidizing influence of the atmosphere, and will not cause scrap in rolling, the latter are decidedly harmful and dangerous, when they are arranged so closely to the ingot-surface that they are affected by the oxidizing influence of the atmosphere, and render the ingot-surface so weak that it cannot withstand contraction, or that it is torn apart in rolling.

These dangerous blow-holes must be prevented, since they are doing all the damage to the ingot, while solidifying in the mold and in rolling.

An ingot, containing a broad ring of blow-holes close to the skin, will naturally be much weaker here than when the skin consists of solid steel of considerable thickness. The ingot, contracting in the mold, while cooling down, will already be quite cool and solid in the four corners, while the four sides, especially in the middle, are still very hot and much less solid than the corners. Contracting, the ingot will exert a tension on these sides, since it cannot yield in the corners, where naturally great friction exists between ingot and mold. The weaker the solid skin of the ingot has been made by blow-holes, arranged directly underneath, the less it can withstand the heavy pull of the contracting steel, and the easier the ingot-surface will therefore be torn apart.

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The same will happen when the steel, by running between stool and mold, forms a fin of solid metal at the bottom of the ingot. The steel contracts not only sideways, but also vertically. At the bottom the fin prevents the free contraction, while in the upper part the ferrostatic pressure crowds the still plastic steel close to the walls of the mold, causing thereby heavy friction between ingot and mold. The ingot-surface must consequently crack. But even when there is no fin of solid steel at the bottom of the ingot its surface may crack, since the ingot, while pressed against the walls of the mold in its upper part, may hang itself up in the mold, due to the contraction of the lower part. If the ingot-skin is weakened by dangerous blow-holes, very deep and bad cracks may develop. Such cracks generally lie in the upper part of the ingot, and are more likely to occur in "bottle-top" than in "open-top" molds.

Blow-holes which are separated from the outside air by a very thin solid skin only, will often open up in the soaking-pit. Here the already thin skin is still more reduced by the formation of scale on the ingot, while the heating expands the gases in the blow-holes, so that they may break through. Such ingots, coming out of the soaking-pit, will look pitted. Many blow-holes have been opened, and the air, coming in contact with the sides of these blow-holes, will oxidize them. Even when the blow-holes are still covered with a thin layer of steel, the rolling will easily tear apart this thin solid surface, and many fine (often, indeed, large) cracks will appear on the blooms. These cracks, at first straight, will by and by become V-shaped; their sides will overlap; and the finished product will show "snakes" and "flaws." Dangerous blow-holes, lying close beneath the ingot-surface, will therefore be the chief cause of scrap in the rolling-mill and the rejections in rail-, plate- and other mills.

The prevention of such blow-holes, even at considerable expense, will pay for itself by lowering the percentage of scrap and "seconds," and improving the quality of the finished products.

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Zinc Oxide in Iron-Ores, and the Effect of Zinc in the Iron Blast-Furnace.

BY JOHN J. PORTER, CINCINNATI, OHIO.*

(Toronto Meeting, July, 1907.)

UNUSUAL problems have arisen at certain iron blast-furnaces in Virginia through the fact that the ore-supplies, derived from the Oriskany formation, contain from a trace up to 1 per cent. of zinc oxide. Since the technical literature bearing on this subject is very scant, the following notes, based on personal experience, will probably be of value to those of our members who are interested in iron blast-furnace practice.

The course of zinc through the blast-furnace is readily traced. Entering the throat as zinc oxide, finely disseminated through the ore, it descends unchanged to the fusion-zone, since a temperature of 1,000° C. or more is required for its reduction. At the fusion-zone, the zinc oxide is reduced by solid carbon to metallic zinc, which is set free as vapor and rises with the ascending gases to the cooler zones of the furnace. In cooling through the range of temperature from 1,000° to 500° C. the zinc is re-oxidized according to the reaction $Zn + CO_2 = ZnO + CO$, and the resulting zinc oxide, being in a very fine state of division, is readily carried along by the ascending gases. Portions, however, deposit on the descending stock and are carried down, again to pass through the same cycle of changes. Still other portions deposit on the lining of the stack, and gradually form hard masses of "cadmia." Of the remainder of the zinc oxide, the greater part passes out of the furnace, and is either deposited in the down-comer, dust-trap, stoves, boilers and flues, or passes through the chimney into the atmosphere. A small portion escapes reduction and enters the slag as zinc oxide, while still another portion is absorbed into the lining of the furnace.

* Assistant Professor of Metallurgy, University of Cincinnati.

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Although a quantitative estimate of the final distribution of the zinc oxide charged into a furnace is attended with much difficulty and uncertainty, the following approximate results, covering a period of six months at one plant, may be of interest. The figures are percentages of the total zinc oxide charged into the furnace during the period, as determined from the weight of ore used, and the average percentage of zinc contained therein:

	Per Cent.
In furnace-cadmia (weighed),	8
In down-comer deposits (weighed),	9
In flue-cadmia (weighed),	13
In flue-dust (estimated),	9
In dust-catcher dust (estimated),	8
Absorbed in hearth- and bosh-lining (weighed),	4
Absorbed in inwall-lining (estimated),	5
Eliminated in slag (calculated),	12
Lost at furnace-top through the lowering of bell (calculated),	9
Unaccounted for and error, in stoves, boilers, chimney-flues, and lost through draft-stack (difference),	23

The difficulties which arise from the presence of zinc in the blast-furnace may be divided into three classes, according to the causes:

(1) Mechanical action; (2) physical action; and (3) chemical action.

1. *Mechanical Action.*—Under this head are such troubles as the choking of the stove-checkers and gas-flues by the zinc oxide dust, and the obstruction of the down-comer and furnace-throat by cadmia formations. These formations are of considerable interest, and have been noted in technical literature.[1] They form a ring adhering to the lining just below the point where the stock strikes in charging, which frequently attains sufficient thickness to obstruct seriously the passage of the stock and gases. In furnaces having a single point of take-off for the gases the tendency is for the cadmia to deposit chiefly on the side opposite this point, which destroys the symmetry of the stock-line and causes an uneven distribution of the stock, with all its attendant evils.

[1] Taylor, A Blast Furnace Problem with Zinc, *Engineering and Mining Journal*, vol. lxvii., p. 469 (1899); Means, The Flue-Dust of the Furnaces at Low Moor, Va., *Trans.*, xvii., 129 to 131 (1888–89); Firmstone, Note on a Deposit of Cadmia in a Coke Furnace, *Trans.*, vii., 93 to 99 (1878–79).

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The cadmia has a stratified structure, is greenish gray in color, is exceedingly hard, heavy and tough, and closely resembles the mineral zincite in all physical properties. It usually contains from 85 to 90 per cent. of ZnO, and from 0.5 to 1 per cent. of metallic zinc.

In the early days of the iron industry of Virginia, these cadmia deposits were not recognized as having any value, and large quantities were thrown on the dumps. The application of chemistry to blast-furnace practice, however, caused the value of the cadmia to be recognized, and now it is considered an asset of sufficient importance, when recovered, to offset largely the difficulties attending its presence in the furnace.

The prevention of the difficulties due to mechanical action is quite simple, and for the most part very effective. Two-pass stoves, with wide checkers and overhead flues, practically eliminate any troublesome clogging. A down-comer of ample size and proper inclination will usually remain free from obstructions during an average blast, while if an obstruction should form, a stop of a few hours will suffice to cut out the cadmia at the throat, where it deposits the thickest.

The removal of the cadmia formed in the furnace itself is more difficult. The old-time remedy was to blow out the furnace every six months and remove the deposits. A variation practiced by some adventurous furnace-men was to bank the furnace and cut out the cadmia from a swinging scaffold, but in this case it is needless to say that the work was not accomplished without the "gassing" of every one employed on the job.

In recent practice encouraging results have been obtained by the use of cast-iron stock-linings. The zinc oxide apparently does not adhere to the iron with any considerable tenacity, and any thin coating that forms is cracked off, leaving the plates clean. Water-cooled plates also have been tried, but they are no more effective than the plain ones, while they are quite liable to crack and allow the water to leak into the stack and backing. In one case this leakage caused a serious scaffold, the cause of which remained unsuspected for some time.

One result of the cast-iron stock-lining, which was not anticipated, was the formation of a cadmia-deposit on the brick lining lower in the furnace. It had been thought that the wear

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of the stock would prevent any accumulation there, but in one instance which came to my notice a zinc ring, about 1 ft. thick and 3 ft. wide, was found at this point after a blast of nine months. The deposit, however, was much smaller than would have formed if the iron plates had been omitted; moreover, its thickness was nearly uniform, so that there was but little effect on the distribution of the stock.

2. *Physical Action.*—Under this head are included the disturbances due to the absorption of heat by masses of zinc oxide which reach the hearth. These masses may come either from a cadmia-deposit at the stock-line, or from a scaffold or other accumulation lower in the furnace. In explanation of this latter assertion it should be said that zinc oxide possesses the power of entering into these accumulations to an almost incredible extent, and greatly augments the difficulties due to them. It is not generally so recognized, but I believe that zinc oxide not only enters into the scaffold after its formation, but also very greatly increases the tendency to form these accretions; my opinion being based largely on the great tenacity with which zinc oxide adheres to the lining.

In an earlier article on this subject,[2] I took the ground that a "zinc-slip," or fall of zincy material to the hearth, would have no material effect on the working of the furnace, my opinion being based on calculations of the theoretical heat-absorption caused by the fall of one ton of cadmia-deposit. Later experiences have caused me to modify this view to some extent. It is probably true that a fall of cadmia-deposit coming at a time when the furnace is hot will cause only a slight chilling, and will not, as a rule, affect the grade of the iron. The fall of a scaffold or scab containing a large proportion of zinc, however, is a more serious matter, since the heat demanded for the reduction and volatilization of the zinc is taken from the hearth of the furnace at the time when it can least be spared.

After a heavy slip of this character the cinder comes up black and glassy, fumes strongly of zinc oxide, and shows little blue flames of burning zinc as it runs down the gutter. The gas at the top of the furnace when the bell is lowered burns with a white flame and gives off dense yellow fumes of zinc

[2] *Iron Age*, Mar. 24, 1904, p. 10.

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oxide. At some of the older furnaces, having badly-cracked crucible-jackets, it is not uncommon to find metallic zinc oozing out of the cracks at such times.[3] The change in the condition of the furnace takes place very suddenly; and, as a rule, the greater portion of the zinc is eliminated from the hearth within 3 or 4 hr. There is usually a second less-marked appearance of zinc in the hearth about 10 or 12 hr. after the first one, and occasionally a third after still another interval. These I attribute to the deposition of a portion of the zinc as oxide upon the stock at a certain zone in the furnace, whereby it is returned to the hearth. I have never been able to detect zinc in the cast-iron made during these periods, and it enters the slag to the extent of only a few hundredths of 1 per cent., or but slightly in excess of the normal amount. This seems rather remarkable, and I am at a loss to explain it satisfactorily.

The cooling-effect exerted by the zinc oxide in the hearth of the furnace is made up as follows: heat necessary to raise the temperature of the zinc oxide from, say, 800° C. to 1,300° C.; heat absorbed by the reduction of zinc oxide by carbon; latent heat of fusion of the resulting zinc, and latent heat of evaporation of the zinc. (Since the subsequent condensation and reoxidation of the zinc take place in the higher zones of the furnace, the heat developed is not available in the hearth and need not be considered here.) Based on a quantity of 1 lb. of zinc oxide, the loss of heat in pound-calories in the items just enumerated is:

	Calories.
Sensible heat absorbed, 0.15×500,	75
Heat of reduction, $1051 - 356$,	695
Latent heat of fusion,	22.5
Latent heat of evaporation (approximately),	425
Total,	1,217.5

Assuming that 1 lb. of coke in the blast-furnace develops 3,800 calories, it is evident that 0.32 lb. of coke will be necessary to furnish the 1,217.5 calories, while an additional 0.148 lb. of carbon, or, say, 0.165 lb. of coke, is consumed in the reduction reaction, making a total of 0.485 lb. of coke for 1 lb. of zinc oxide.

[3] See, also, Firmstone, Note on a Deposit of Cadmia in a Coke Furnace, *Trans.*, vii., 93 to 99 (1878–79).

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In order to prevent any possible misunderstanding, it should be noted that, while these figures refer to the heat lost to the hearth of the furnace, the calculations given in my former paper, previously referred to, deal only with the heat lost to the furnace as a whole. This is the chief cause of the considerable difference in the two results.

The only method which has been developed to correct this disturbance is the crude one of charging extra fuel upon the first indications of a "zinc-slip." Probably in this case "an ounce of prevention is worth a pound of cure," and extreme care to prevent the formation of scaffolds would be of more benefit than anything else. It is unfortunate, therefore, that the poor blowing-equipment and the absence of proper stocking-facilities at many of the plants in Virginia cause them to run irregularly, and render them particularly liable to this form of trouble.

3. *Chemical Action.*—Under this head may be discussed the action of zinc oxide on the fire-brick lining of the furnace. It is unusual for a furnace in this district to run more than a year without relining the hearth and bosh, and although not generally so recognized, I believe that the presence of zinc oxide is the chief cause of this deterioration. An examination of the lining remaining in these furnaces after blowing-out reveals the following facts: The bricks in the hearth and lower part of the bosh are very soft and of a greenish-black color. They contain considerable carbon, and 40 per cent. or more of zinc oxide, while small yellow crystals of the zinc oxide are abundant. Higher up in the furnace the bricks are firmer, and contain less zinc oxide and carbon, while above the mantle the structure is unchanged, but the color is deep blue. Analysis shows only a trace of carbon, with about 20 per cent. of zinc oxide. The blue color is commonly attributed to zinc by the furnace-men of the district, but as I have noticed it in other furnaces not using zincy ores, I am disposed to attribute it to the presence of titanium in the brick. The researches of Dr. Seger have shown that titanic acid gives a blue color to kaolin when heated to a high temperature, and it is well known that many, if not most, of our fire-clays contain appreciable amounts of this element.

Dr. Steger, in an investigation of the cause of the disintegra-

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tion of zinc-muffles,[4] found that the zinc-vapors attack the clay substance according to the reaction $Al_2O_3, 2SiO_2 + Zn + CO_2 = Al_2O_3, ZnO + CO + 2SiO_2$, forming zinc-spinel and tridymite. Willemite ($2ZnO, SiO_2$) is also formed. Dr. Muehlhauser finds that the formation of zinc-spinel takes place principally when the muffle is still new and porous. The absence of a glaze on the side next to the fuel allows the gases to diffuse into the walls of the muffle, where, meeting the zinc-vapors, the reaction takes place.

The work of Dr. Steger and Dr. Muehlhauser affords a very clear explanation of the action of zinc on the iron blast-furnace lining, although it does not account for the presence of finely-divided carbon or of free zinc oxide. This latter, however, probably results from the fact that the lining is considerably cooler than the hearth-space, permitting the oxidation of zinc by carbon dioxide. It is assumed that zinc-spinel forms at temperatures above the reduction-point of zinc oxide, which is more than 1,000° C., and, this being true, the clay would be attacked on the interior surface of the lining, with a formation of zinc-spinel, while farther out, where cooler from radiation, there would be deposition of zinc oxide without disintegration of the brick. It is actually found that the bricks on the inner surface of the furnace are badly disintegrated, have lost all trace of the original structure, and are so soft that they can be crumbled between thumb and finger, while farther back the structure of the brick begins to show; it is harder, and the yellow crystals of zinc oxide are abundant. These observations show that theory and fact are in close agreement.

So far as I am aware, no experiments have been tried with the object of reducing the action of the zinc on the lining of the iron blast-furnace. It is improbable that anything could be done with a glaze, since the wear of the lining is comparatively rapid. A series of experiments to show the relative resistance of the various brands of fire-bricks to the action of zinc-vapors would undoubtedly lead to beneficial results, and, in view of the possibility of lengthening the life of the lining, and thereby securing a longer campaign for the furnace, it seems well worth while to undertake the experiments suggested.

[4] *Tonindustrie Zeitung*, abstracted in the *Transactions of the American Ceramic Society*, vol. vii., p. 277 (1905).

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Secrecy in the Arts.

BY JAMES DOUGLAS, LL.D., NEW YORK, N. Y.

(Toronto Meeting, July, 1907.)

THOUGH liberality is not supposed to be a prominent trait of the Scottish character, Canada owes to a Scotchman, Sir Wm. Macdonald, more than to any other of its people, not only wise ideas, but pecuniary help towards extending education: and another Scotchman, in New York, has sumptuously housed under one roof three branches of the engineering fraternity, of which we are one.

Having thus created us members of the same family—for the metaphorical meaning of house and kinship is identical—Mr. Carnegie expresses more emphatically than even he could in words, that, by affording facilities for closest intercourse, he invites the mechanical, electrical and mining engineers to participate in the freest interchange of idea and experience, and to correlate and combine the results of their studies and activities; and, being members of the same household, to banish reserve and secrecy.

And now we, the members of the American Institute of Mining Engineers, meet as though we were at home in a neighboring country. For whether we meet in Canada, in England, or elsewhere, the Institute is always received as though no political or geographical divisions separated its members from those of congenial associations in the land of its host. In truth, the title which we have assumed claims for the sphere of our activities the whole American continent, regardless of such trifling details as boundary-lines. The fact that we meet to communicate each other's experience, to discuss our difficulties, and to seek each other's aid in solving the intricate problems that so often present themselves in the course of our professional life, is an acknowledgment of our individual helplessness; and therefore an argument for united effort. But no effort can be of any value if there is an underlying suspicion

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of reserve and lack of candor in our treatment of the technical questions which it is our province at these meetings to discuss.

Yet there are limits to the extent to which we may go as officials of public companies. We know that even as professional men it is not always easy to reconcile principles with practice, and on this subject of sincerity and transparent diffusion of our experience there is some slight difference of opinion and difficulty of application. Few of us are as favorably situated as the college professors, whose first duty is to unbosom themselves to their students of all they know, and perhaps of a little which they only suppose they know. The most of us are paid officials of corporations whose *raison d'être* is to make money, and whose executive officers sometimes, not without some reason, consider their trade secrets as part of their capital. Some companies confide these secrets to the honor of their technical workers under as strict rules as those imposed on their cashier in the distribution of their money. Unless, therefore, our employers permit, we as employees are under pledge of secrecy. Many large manufacturing, mining and metallurgical concerns put no restriction upon the freedom of their technical staff, but some of our largest certainly impose on their employees absolute silence as to all that transpires within their laboratories or workshops.

If the question were left to us alone, it would be easy of solution. Our reliance on one another, as workmen in such distinct branches of engineering as civil, mechanical, electrical, and mining, is so close that we must co-operate in every large enterprise we undertake. We cannot succeed singly, for few of us claim to be so encyclopædic in our knowledge or universal in our experience as to make it safe to rely on our own acquaintance with the practical details of every one of these important departments. We therefore seek each other's assistance, and consequently share in each other's secrets; for every man's special knowledge is to the other man, who is more ignorant or inexperienced, a secret, or a "mystery," as old tradesmen and professional men used to call it.

Every industrial advance brings us closer together and makes it more impossible to act independently. Till very recently the civil engineer surveyed and located the line of railroad; the metallurgist made the rails and the iron and steel for the loco-

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motives; the mechanical engineer designed the equipment, and what was left for the electrician to do was to string the telegraph wires. Now all this is changed. The electrical engineer is, in importance, springing into the first rank; and he requires such special acquaintance with the mysterious forces, whose distribution and useful energy he handles with so much audacity, and yet safety, that few of us are particularly anxious to meddle with his operations.

Or, to take a mining instance, in reaching a decision as to the adoption, in underground or overground haulage, of electricity or compressed air, even in metalliferous mines, the verdict must depend on so many delicate and purely technical considerations that few superintendents or general managers would consider themselves sufficiently informed to commit themselves until their mechanical and electrical advisers had marshaled all the facts and arguments for and against each system. I need not cite other instances in which co-operation is demanded in the carrying-out of almost every modern enterprise, or in the equipment of the complicated mechanisms which have replaced the simpler contrivances of our forefathers. The modern steamship exceeds Columbus's caravels in size and complexity of construction as much as does a limited train an old stage-coach. And in proportion to the multitude of their parts and the tremendous energy of the forces which must be called into play to move them, is the diversity of knowledge, talent and skill required to design, construct and operate them. As I have said, therefore, if it depended upon us alone, there would be no difference of opinion as to the necessity of perfect open-mindedness, and as little backwardness in applying this principle to practice.

I think, moreover, that many of us are also convinced that open-mindedness to the suggestions of others is a useful quality to carry into our work, even after we and our scientific staff, in our joint wisdom, have co-operated in formulating and laying out our plans. Every draftsman in our offices, every master mechanic in our shops, every foreman in our mines, is an expert in his particular line, and may be supposed to be familiar with minutiæ which have escaped our observation, or, to put it frankly, may know more than we do ourselves on some of the innumerable details which make up the sum total of the

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questions on which we have ultimately to pass. The suggestions coming from such subordinate sources may not always be worthy of acceptance; but, on the other hand, it is never wise on our part to turn a deaf ear or a disdainful shoulder to them. In our own small way and work, we feel sometimes almost ashamed to accept credit accorded to us for ingenuity and foresight in devising either mechanical or industrial novelties, or in organizing enterprises, when we recall the hints from others which have suggested thoughts to ourselves. For though undoubtedly, especially in the most imposing cases, the one inspiring mind has conceived the idea, either of the invention or of the enterprise, only by the co-operation of a number of subordinate agents, who often belong to the multitude of the unknown and forgotten, could the idea have been worked out to the glory and the profit of the inventor or promoter. It is always difficult to decide how much of the result should be passed to the credit of the conspicuous man and how much to that of the hard-working, plodding, unimaginative grubber among details, or to the brilliant, erratic, and, because erratic, unsuccessful fellows who do so much of the world's work, and yet get so little of the praise or the gold. But this is certain, that in our own work, whether it be conspicuous or not, we shall always benefit by inviting suggestions from the humblest of our co-operators, encouraging them to think independently, and fearlessly to express their thoughts. That this is already one of the prominent qualities of American industry is manifest from the large number of patents which are taken out by inconspicuous laborers in the fields of engineering, especially of mechanics.

I recently went, one Friday morning, into our purchasing department and found every desk empty, and all the clerks assembled in the manager's room. He had found it conducive to the most efficient conduct of his office to assemble all his staff once a week for free discussion as to the most economical distribution of work. The result was that instead of late hours and overtime the necessary day's work was obtained very easily within office hours; for suggestions from the men actually engaged were found to be worth more than those from the men merely superintending. The youngest were not snubbed, if they ventured to make a remark; and thus the spirit of pride and co-operation pervaded the whole group.

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We have all heard how Mr. Carnegie is said to have stimulated the rivalry of his superintendents and heads of departments by getting them to meet at intervals over an excellent lunch, where around the table all jealousy vanished, and little technical secrets, which are liable to exist even among friends, were revealed. But good cheer is not, or should not be, necessary to make the great brotherhood of technical workers unfold their sympathy and unbosom their secrets to one another.

But we return to our main subject. How far and how completely should even corporations and industrial concerns permit and encourage the interchange of information? I am inclined to think that few limits should be set; for every limitation means the concealment of some fact or some principle which only if revealed can be developed to its full significance and utility. As a rule this can be best done by the action of many minds and many hands. Till developed it does not yield its full advantage to even the original discoverer, for he alone, unaided, in the solitude of his laboratory or behind the bars of his factory, without the practical assistance of his fellow-workers, rarely brings his original germinal idea into efficient, practical utility.

Were we free to appeal to purely altruistic motives, it would therefore be superfluous to argue in favor of complete knowledge- and experience-sharing, but profit-sharing is after all the impelling motive of industrial advance to-day, as it has been in all ages, and to reconcile the admitted evils of secrecy with the admitted advantages of publicity, the patent laws have been framed. They have always given the patentee the right to use either in his person or through his agent his invention or discovery for a limited number of years, provided he describes it so fully that it can be practiced by one skilled in the art. The publicity and knowledge conveyed by the specification stimulates the inventive faculties of others, and patented and unpatented improvements, along the line of the original invention, demonstrate both the value of publicity and the cupidity of men, even of the technico-scientific class. Among the great inventors of our day was Sir Henry Bessemer. Before the Royal Commission, appointed to suggest revision of the British patent laws, he gave evidence, some of which he has embodied,

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with very suggestive comments, in his autobiography.[1] Before he became famous he devised machinery for making bronze powder, and manufactured the powder in secret. Of this he says:

"While referring to patents for inventions, I cannot refrain from pointing to this particular invention of bronze powder as an example that may advantageously be borne in mind by those short-sighted persons who object to grants of letters-patent. There can be no doubt of the fact that the security offered by the patent law to persons who expend large sums of money and valuable time in pursuing novel inventions, results in many new and important improvements in our manufactures, which otherwise it would be sheer madness for men to waste their energy and their money in attempting. But in this particular case the conditions were most unfavourable for patenting, owing to the fact that the article produced was only a powder, and could not be identified as having been made by any particular form of mechanism. Therefore it could not be adequately protected by patent; moreover, by my machinery, the cost of production, if only paid for at the ordinary rates of wages, did not exceed one-thirtieth of the selling price of the article. This fact alone offered an irresistible temptation to others to evade the inventor's claims, and so rendered the patent law a most inadequate protection. On the other hand, the great value of a small bulk of the material made it possible to carry on the manufacture in secret, and this method of manufacture was rendered the more feasible by making each different class of machine self-acting, and thereby dispensing entirely with a host of skilled manipulators. It may therefore be fairly considered, so far as this particular article was concerned, that there were, in effect, no patent laws in existence.

"Now let us see what the public has had to pay for not being able to give this security to the inventor. To illustrate this point, I may repeat the simple fact that the first order for bronze powder obtained by my traveller was for two pounds of pale-gold, at eighty shillings per pound net, for the Coalbrookdale Iron Company. I may further state that, in consequence of the necessity for strict secrecy, I had made arrangements with three young men (my wife's brothers), to whom salaries were paid far beyond the cost of mere manual labour (of which, indeed, but little was required). My friend Mr. Young desired to occupy the position of sleeping partner only, and not be troubled with any details of the manufacture; so I entered into a contract with him to pay all salaries, find all raw materials, pay rent, engine power, and bring the whole produce of the manufactory into stock, in one-ounce packages, ready for delivery, at a cost, for all qualities, of five shillings and sixpence per pound; after which he and I shared equally all profits of the sale. It is rather a curious coincidence that the one-ounce bottles of gold paint were labelled five shillings and sixpence each, off which the retailer was allowed a liberal discount.

"Had the invention been patented, it would have become public property in fourteen years from the date of the patent, after which period the public would have been able to buy bronze powder at its present market price, viz., from two shillings and threepence to two shillings and ninepence per pound. But this important secret was kept for about thirty-five years, and the public had to pay excessively high prices for twenty-one years longer than they would have done had the invention become public property in fourteen years, as it would have

[1] *Sir Henry Bessemer, F.R.S.: An Autobiography*, pp. 82 to 85. London: Offices of *Engineering* (1905).

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been if patented. Even this does not represent all the disadvantage resulting from secret manufactures. While every detail of production was a profound secret, there were no improvements made by the outside public in any one of the machines employed during the whole thirty-five years; whereas during the fourteen years, if the invention had been patented and published, there would, in all probability, have been many improved machines invented, and many novel features applied to totally different manufactures.

"I have lingered long over this subject of bronze powder, because it is one which has had great influence on my career; it was taken up at a period when my energy and my endurance, and my faith in my own powers, were at their highest; and as I look on all the incidents surrounding it, through the lapse of time and the many changes of the fifty years since it was undertaken, I wonder how I had the courage to attack a subject so complicated and so difficult, and one on which there were no data to assist me. There were not even the details of former failures to hold up the finger of warning, or point out a possible path to pursue, for no one had yet ventured to try and replace the delicate manipulation which experts had made their own, both in Japan and China, where texts and prayers printed with bronze were offered up at the shrine of Confucius two thousand years before I had ever seen a particle of bronze powder."

He concludes this first reference to his powder in the following paragraph:

"In closing these details of the bronze powder manufacture, I may say that, later on, the handsome royalties paid by my steel licencees rendered the bronze powder business no longer necessary to me as a source of income; and I had then the extreme satisfaction of presenting the works to my brother-in-law, Richard Allen, who had, with so much caution, successfully kept, for more than thirty years, a secret for which, he perfectly well knew, some thousands of pounds would have been given him at any moment."

But he returns to the subject of patents when discussing another invention of his, that for making optical and plate glass, the value of which, for some reason or other, the trade has never appreciated. He says (pp. 117, 118):

"There is one point in connection with patented inventions upon which I have always felt strongly. I have maintained that the public derive a great advantage by useful inventions being patented, because the invention so secured is valuable property, and the owner is necessarily desirous of turning that property to the greatest advantage; he either himself manufactures the patented article, or he grants licences to others to do so. In either case the public reap the advantage of being able to purchase a better or cheaper article than was before known to them, due to the inventor's perseverance in forcing his property upon the market. But if a novel article or manufacture is simply proposed by a writer, and published in the technical press or in newspapers, as a rule (almost without a single exception) no manufacturer will go to the trouble and expense of trying to work out the proposed invention. He says to himself: 'I shall not risk the expense necessary to develop this new idea, for it may entirely fail; or even if I succeed, its development will cost me much more than it will cost other manufacturers, who will immediately avail themselves of it if I succeed; no, let some one else try

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it;' and so the invention is lost to the world in consequence of having been given away. This loss to the public is equally the case with patents that are not taken up; and one of the simplest and most effective inventions which I have ever made may be here cited as an example, as it formed part of the novel system of plate glass manufacture just referred to."

After describing his plate-glass invention and its public neglect, he says (p. 122):

"From what I have said I think I have shown that, however self-evident an invention may be, or however advantageous it might be to a manufacturer, if it is public property he will not touch it."

Sir Henry was doubtless correct in asserting that under the impulse of self-interest inventions are pushed by the inventor more vigorously than if he had merely the scientific credit due and given to investigators to spur him on. But he really gives us an argument against patents when he describes the apathy of the public to his glass patents. His steel patents he worked out himself and brought to perfection after years of heavy expense and labor, and by adopting certain modifications to fit special cases. His plate-glass patents he never himself applied, but they would probably have been loaded down with royalties which the trade did not care to pay while taking the risk of applying them to practice. As to his bronze powder, he would probably have made far more out of it had he reduced the exorbitant prices and increased the demand, even while manufacturing it behind closed doors.

It is, however, foreign to my purpose to discuss the patent laws, except casually as they bear upon the subject of secrecy in the arts. Sir Henry's generalizations are substantially correct, but they are too sweeping; for there have been great inventions which the public has not been backward in using, though they were freely given to the world. One's thoughts pass with pleasure from the contemplation of the money-making inventors and investigators to such prophets and apostles of science as Sir Michael Faraday. Sir Michael's profound and original investigations into electricity and magnetism gave the world the dynamo. Though he did not work out the mechanical details of a practical generator, he undoubtedly invented appliances which might have been used for making a strong basis-claim for a patent. But nothing could have been more repulsive to his spirit or foreign to his high aim in life than gauging his time and talents by a mere money standard. He

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lived contentedly on the small salary he received from the Royal Institution, preparing his lectures to children with as much care as he bestowed on those delivered before the Institution which made him famous; and turning his great learning and power of investigation to the nation's good in return for very scanty remuneration, for he deliberately decided to devote his life to scientific research for truth's sake, rather than to use his vast attainments in the service of Mammon. At the commencement of his career, Faraday added to his salary from the Royal Institution by what he called commercial work. At first, his average earnings from this source were £240 per year. By 1831 they reached £1,090. By 1838 they had shrunk to nothing; for in the meantime his great discovery of magneto-electricity was made, and his thoughts were so intently directed to his experimental work that no time could be spared for money-making.

Faraday's mind was too absorbed in wonderment and almost religious fervor, as the secrets of nature revealed themselves, for sordidness in any form to find lodgment. He wasted his energy neither in money-making nor in captiously defending his discoveries and great conceptions from supposed infringement by other scholars. He never forgot, despite his brilliant original work, that generally the great investigators only lay the keystone in the arch which many less gifted workers have been erecting stone by stone. He did not consider it any detraction from his honor that he was permitted only to crown the structure which others had helped to build from the foundation up. They are the greatest among the great who appreciate this limitation and recognize what they owe to others. Faraday knew that his discoveries gave him but imperfect glimpses of some of the laws and phenomena of nature, which we, through our ignorance and prejudice, are slow in understanding, but which would soon cease to be secrets if we could only disabuse our minds of false conceptions, see facts as facts instead of as arguments for our theories, and then work together with single-heartedness.

Faraday felt also, as every true disciple of science should feel, that when we penetrate to the discovery of even the least important of the facts of nature, we are unveiling one of God's gifts to humanity. If that be so, we may well ask ourselves, what right have we to draw the curtain over it and conceal it again from God's children in order that we may make money

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out of it? Looked at from this point of view, may we not question the right to buy men's thoughts; and when through their thoughts we have unraveled some of nature's secrets, and learnt something to our profit, use our power and another's necessities to impose secrecy? Are we not enslaving a human mind, and can any slavery be worse? A natural phenomenon of force, once so understood that it can be controlled,—is it not as much a gift of God as rain and sunlight, and therefore part of the heritage of all mankind? Corporate wealth and corporate energy are doing much for mankind, despite the fact that of corporate shortcomings we hear just now more than we do of corporate benefactions. But corporate influence will have reached its most beneficent development when the wealth and activity and masterful management of the able men who wield it are united to the knowledge and skill of their technical staff in not only discovering but publishing the truths of nature, which they may be agents in revealing. Nor are we oversanguine in believing that this high aspiration, if carried into practice, would not interfere with the lower motive of their existence, money-making.

Sir Michael Faraday is not the only worker in the field of practical science who has given the results of his labor unreservedly to the world. It is almost invidious to single out instances when so many distinguished and such a multitude of obscure toilers are working at the intricate problems of technology from sheer attachment to truth and without any thought of gain. But three notable names may be mentioned as representative of this noble army of the unselfish—our own Prof. Henry, Dr. Roentgen, and that devoted couple, whom we rank as one—for, as husband and wife, they were as united in love one of another as they were one in love of science—Mons. and Madame Curie. The Roentgen ray may have needed no patents, or patentable devices, or any business organization to push its beneficial applications, especially in the alleviation of suffering humanity, but it would not have been difficult to concoct patents had Dr. Roentgen, before describing his discoveries, wished to make money out of them. And could the practical resources of radial activity be measured by dollars, what a fortune the bereaved widow would reap! But Prof. Roentgen enjoys a better harvest than royalties, and Madame

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Curie would not exchange for a mountain of gold the world's admiration and reverence for her husband's memory.

But to descend to a lower plane. If it is the fact that technical science has progressed of late with such unwonted speed through the co-operation of many workers, and that this co-operation has been made possible by the publication and exchange of ideas and experiences in the technical and scientific journals, would not our progress be even more rapid and thorough if all barriers of secrecy were broken down, and every encouragement were given to our technical workers to describe, in print and by conference, their notions and their actual experiments? This is the attitude of some, I may almost say of most, of our large concerns, but unfortunately it is not that of all. It is impossible to compare, as to efficiency and profit, works the gates of which are fast shut, and in which obscurity and secrecy are imposed and practiced, with those to which free admission is granted and in which freedom of information is encouraged. But the following reflections force themselves upon us in this connection. We know that very few technical papers issue from certain establishments; that on their officials silence is imposed; and that to these works inquisitive visitors are politely but peremptorily refused admission. There are not many such, but they are and have been very successful. But suppose that in imitation of their practice and regulations all were tempted to adopt it, so that the same policy became univeral; what a sudden paralysis of industry would follow! Our secretaries would find it difficult to fill even their shrunken volumes of transactions with papers worth printing; our students would have to content themselves with the antiquated learning which their professors could supply; for there would be no more summer classes for practical work in mines, smelters and electrical factories, and the professors themselves would have to learn from old books. Every manufacturer and smelter would be obliged to bribe his neighbor's workmen and tempt away his neighbor's superintendents for information. As a result, before long, the very works which now find it so profitable, or think they do, to tap their friends' stock of knowledge and experience, and give nothing in return, would be driven in upon their own resources, and would undoubtedly then find them not so complete as they imagine.

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Of course, I am supposing an impossibility, because the spirit of intellectual freedom in our professions is too strong and too widespread to submit to such a tyranny, and because, before such darkness of ignorance had settled down on our great industries, the most pronounced advocates of secrecy would feel and acknowledge the ultimate consequences of concealment, and would become reformers. To-day they may have secrets, as valuable as Sir Henry's method of making plate glass and bronze powder, which it may pay them to conceal from their competitors, so long as they are admitted freely to their competitors' open shops; but even this is doubtful. For the spirit of secrecy is intimately allied with the spirit of suspicion and distrust; and the mind which is always suspecting is closed tight against the admission of fresh and fair impressions. Being jealous of others, it is prejudiced against their suggestions, and correspondingly prejudiced in favor of its own preconceptions. Progress therefore ceases.

This is a temper of mind foreign to a new country like ours, whose special industries have not been established long enough to wear grooves of rigid practice and sink into ruts of self-satisfied indifference. About the best correction we can apply to the growth of dry-rot is the banishment of secrecy. A curious instance of its blighting influence is seen in some of the older, not the newer, industries of the old world. The iron- and steel-works of Europe have not kept pace with ours in size and production, but the ironmasters of Great Britain and Germany, in coke-making and in blast-furnace economies and in steel-making processes, have been our teachers. Nor have they been shy of communicating their improvements, or, through jealousy of our success, slow in adopting ours. No nobler monument of international comity in thought and experience exists than the seventy volumes of the *Proceedings of the Iron and Steel Institute.* And with few exceptions the iron- and steel-works of England, Scotland, Germany and France are open to any accredited worker in the same domain. Yet before England was conspicuous as a maker of iron, she was famous the world over for her copper- and tin-production. But, between self-conceit and the inbred habits of trade-secrecy, her copper-smelting industry has fallen from its high estate. And it is not accidental, but linked as closely as any effect with its cause, that this decline is in great part the result

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of habits of secrecy which grew with the growth of age. At Swansea, every gate to the smelting-works is guarded, and as a result it has been as difficult for thought to escape out as for suggestions to find their way in. Swansea should still enjoy the leadership which her skilled labor, splendid coal and commanding maritime situation put within her reach; but she has preferred to gloat over her secrets behind closed doors rather than go out into the world in search of new business as well as technical methods, while also inviting the world to enter and exchange ideas with her. What is the consequence? New Zealand copper comes here to be refined, notwithstanding the first practical application of electrolysis to metals was made by Elkington in England, and the Vivians adopted the Manhès method before Farrel introduced it into this country.

There are, however, of course, exceptions in England to this too prevalent habit of secrecy. To the works of the Rio Tinto at Port Talbot or of the Cape Copper Co. at Briton Ferry in South Wales, where metallurgical novelties have been tried, introductions are not refused. But the alliance of decay and suspicion in the instance I have given can hardly be accidental; and we may be sure that what is baneful in its effects in Europe is not likely to be beneficial here; for while the Atlantic separates continents it does not delimit the operation of laws.

In political life, vitality is maintained only when every man takes his full share as a debater in the discussion of political questions, and as a voter in the determination of state affairs. So in scientific and technical matters, the banishment of deceit, mystery and jealousy, and the freest admission of daylight by means of the unreserved diffusion of information through the press and personal intercourse, will instill into the whole body of workers a feeling of healthy rivalry, which, while stimulating their mental activity, will correspondingly benefit the financial interests of their employers.

I have supposed an extreme case—that the example set by our few secretive establishments were followed by all. Let me imagine a more probable issue, such as, I believe, will result from the fellowship of knowledge and experience which Mr. Carnegie, in presenting to our national engineering societies their new home, urges them to cultivate—namely, that all our technical manufacturers will learn how they gain, and not lose,

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by encouraging their staff-officers to study their neighbors' methods, and by throwing open their own establishments, in turn, to the freest criticism of their competitors in trade. What will result? Nothing but advantage, I believe, to all whose wisdom and means have enabled them to provide themselves with the raw material of manufacture on advantageous terms, and to locate their works or factories at localities favorable for economical operation. Loss only to those who, in any case, ought to go out of business, because they have failed to secure the conditions essential to success! And, above all, benefit to the public, which, after all, is the finality we should always keep in view.

How, now, can these two cardinal conditions—financial success and public approval—be best attained? Unquestionably, by mutual help and the most unreserved publicity. In any branch of industry, no intelligent worker claims that he and his staff have attained either the utmost economy in operation or the most thorough acquaintance with all the reactions which enter into the processes which he practices. Each knows that hundreds of other intelligent and well-informed men are eagerly at work on the solution of the same problem. Some may be a little cleverer than others, and some may have made a little more progress in certain lines than their co-workers. But this discrepancy will not necessarily continue; for the clever fellow is picked up by rival works, the secret so carefully guarded leaks out, and the disturbed average of paid ability and of stock of knowledge is restored. But if the companies and their staff are unwilling unreservedly to pool their knowledge and experience, the advantage of making into one great stock such accumulated experience and knowledge of these hundreds of workers is forfeited. With certain reservations, and by special permission, many of our larger establishments, in all or in certain departments, are freely open to each other's technical officers; but instead of being admitted upon sufferance, they should be invited in, with full liberty to study processes and test machinery; for assuredly the host would benefit as much as the guests by the discussion which would follow such unreserved exchange of ideas and comparisons of appliances and methods.

I have referred to certain limitations to publicity. One, un-

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doubtedly, is costs. Under our present economic system, no manufacturer or miner or metallurgist cares to give away his costs, and that for very obvious reasons. What they are may be inferred, but professional courtesy forbids direct inquiry into that delicate subject. This restriction, however, need not interfere with unstinted technical freedom of intercourse. There is, moreover, another judicious limitation to publicity. Most of our largest concerns are incorporated and financed as joint-stock organizations, in which thousands of technically ignorant and helpless shareholders are interested. Unquestionably, indiscriminate admission to works and mines must be refused, for, unfortunately, there would be visitors who, if admitted, after the visit would tell remarkable stories, from actual observation, with the view to affect the value of stocks. But such restrictions do not affect the main proposition that mercantile concerns of every class, depending for success on technical knowledge and skill, would gain by the removal of restraint on the thought and action of their technical staff.

I am not blind to the fact that the same object is sought to be attained by the consolidation of many works under one organization, or by the encouragement of friendly financial co-operation among even competing companies; but this tendency to consolidation has not yet succeeded in obliterating competition, and will not as long as there are active, intelligent men among us, who prefer to rule rather than to be ruled, and to manage their own business rather than have it managed for them.

On the benefits or disadvantages of the present movement towards consolidation of works and the combination of capital in large industrial undertakings, there is, and will be, of course, considerable diversity of opinion. That competition is wasteful and is encumbered with other evils few will deny; that it has a keenly stimulating effect all will admit. Yet it remains to be determined whether a board of absentee managers and paid officials will be a compensating substitute for the ambitions, personal pride and tireless energy and skill of the individuals who have built up great works which they may have seen, perhaps reluctantly, absorbed into a combination. There are, apart from the political and sociological aspects of the present consolidation tendency, technical and economical conditions which force themselves upon the consideration of those of us under

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whose management works have grown from small to large dimensions. The difficulty of maintaining a high standard of quality as the demand is made for enormously increased production is urgently presenting itself both to the management and to the public. And it is doubtful whether, after expansion has reached the point where administration charges become light, there is actual economy in unlimited expansion; and whether the most skillful and closely managed corporate organization can replace the personal supervision of a single mind. But what immediately concerns us in the present discussion is the dangerous temptation to adopt secretive methods by very large corporations.

The larger the combination grows the more sensitive will the management be lest business and trade secrets which they possess, or think they possess, be revealed by subordinate officers. The imposition, therefore, of strict rules of silence on all except those in supreme command is likely to result. If the absorption of any one class of our national resources should pass under the control of any one organization, the technical knowledge necessary to the development of that particular resource would be of interest to that organization alone, and the risks of publicity, and therefore the evils of secrecy, would become a merely academic question. This dangerous point, under our present industrial system, will probably not be reached; for state socialism, to which concentration steadily approaches, would be the inevitable alternative and would be adopted before the other alternative had been attained. But it must be to the management of those enormous consolidations a grave consideration how they can give such latitude to the members of their staff as will produce that healthy self-reliance which comes from freedom of speech and freedom of opinion, without endangering the tremendous financial interests for which they are responsible. Whatever individual difference of opinion on this subject there may be among the managers of the great industrial establishments, there is not any difference of opinion in the country at large; and public opinion has to be consulted. Therefore, would it not be safer and better for the interests of the shareholders to adopt the policy of freedom which I have outlined, and thus placate the public? For the growing public anxiety, amounting to animosity and suspicion, against our big corporations would be allayed if it were appa-

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rent that the technical officials of the small concern had at least the right of knowing what the big concern was doing, and the big official did not arrogate to himself the possession of exclusive knowledge and exclusive skill. From the point of view of public policy, the question is one well worthy of consideration; for it is coming about that not only railroads, as public highways, but all large corporations utilizing the country's natural products and converting them into necessary objects of trade, will pass under closer legislative scrutiny and public criticism in the future than they have in the past—a necessary limitation, which will become more exacting the larger the corporations grow—if the tendency to growth continues.

While unquestionably dangers can be foreseen as arising out of these great industrial aggregations—not only of capital but of industrial energy—dangers technical, social and political—there are also great possibilities of good. One of the benefits may justly be claimed to reside in the large funds that are thus rendered available for technical research, from which the public derives benefit indirectly, even if the results are not published. But if we could banish secrecy; if every industrial establishment of any magnitude, which is in its own interest carrying on technical research, should encourage its technical staff to confer freely with the members of every other technical staff, would not the sciences and arts progress far more rapidly than if one huge organization controlled a given industry? All our principal metallurgical and chemical concerns have laboratories, and carry on investigations and make experiments, generally on a large working scale; and surely the advancement of technological science can be better attained in a number of such laboratories than if there were fewer or in only one. There is keener competition of wits when many brains are working independently. The friction of honest rivalry is a force not to be despised. The stimulus of ambition is sure to be stronger in smaller than in large consolidated workshops. The air in such laboratories is freer and purer than when men are working in the stifling atmosphere of secrecy. I believe that such a consolidation of mind and high impulses would carry us further and faster along the road of human progress than all the money that all the trusts could appropriate for the advancement of technical knowledge.

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The Electric-Air Drill.

BY WILLIAM L. SAUNDERS, NEW YORK, N. Y.

(Toronto Meeting, July, 1907.)

MANY members of the Institute, who participated in the visit made, during the Bethlehem meeting of February, 1906, to the shops of the Ingersoll-Rand Company, at Phillipsburg, N. J., inspected with interest the new Electric-Air drill, which the company had set up for the purpose of showing it in actual operation to American mining engineers. At the request of the Secretary of the Institute, I promised at that time to prepare a paper for our *Transactions*, describing the construction and advantages of the machine. But such a paper would then necessarily have contained much that was only expected or claimed by the designers and manufacturers of the drill, and not yet incontrovertibly proved by varied and long-continued practice. However moderate such statements might have been, they would have given inevitably to the paper, to some extent at least, the air of a prospectus, rather than of a technical contribution. I therefore decided, with the Secretary's approval, to postpone the writing of the promised paper until it could set forth the results of adequate actual practice, as well as the latest details of construction, etc., based upon practical experience. That period has now arrived. The Electric-Air drill has been exhaustively tested in the field, under varied and arduous conditions and upon the hardest rocks. It is now fairly in the field; its merits and performances are matters of unimpeachable record, and its place among established competitors can be definitely determined.

As a representative of the Ingersoll-Rand Co., as well as a member of the Institute, I may be permitted to add that my company, being largely interested in the manufacture of air-compressors and machinery driven by compressed air, has no desire to injure its own business by claiming for this new machine that it should immediately supersede all existing applications of pneumatic transmission of power for drilling. On the

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other hand, if we had not satisfied ourselves that it has proved itself the best for given conditions, the company would not

FIG. 1.—THE ELECTRIC-AIR DRILL AND PULSATOR.

have risked its reputation by introducing it, and I, as a member of the Institute, would not have written this paper.

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In former contributions,[1] I have discussed the use of compressed air, and opposed, to some extent, the claims of the advocates of electrical power-transmission in mining. I need not now retract any opinion thus declared. Many features of electrical transmission are undoubtedly convenient and economical; but the direct application of the electric current in rock-drilling has long been a baffling problem; of which, in my judgment, the machine here described has furnished the first, and thus far the only, satisfactory solution, by combining the acknowledged advantages of air-driven percussion with the acknowledged advantages of electric power-transmission, while avoiding the acknowledged disadvantages of both systems.

This drill is correctly designated; it is not an electric drill, but more completely an air-drill than any other in existence, because it can be driven by air only and not, like other air-drills, by steam also. Yet, while it is thus distinctly air-operated, the power-transmission is electric, and the sole connection of the drill with the power-house is made by the electric wire, air-compressors and pipe-lines being entirely superseded.

Fig. 1 gives a general idea of the apparatus. It shows a rock-drill, resembling, at first glance, the familiar air- or steam-driven drill, mounted in the usual way, and doing the same kind of work. Very near the drill, and connected to it by two short lengths of hose, is a small air-compressor, or, more properly, a pulsator, mounted upon a little truck. This constitutes the entire apparatus of a single drill. Each drill is accompanied by its individual pulsator, and each pulsator is connected to the line of wire from the power-house.

The usual drill-shell is employed, and may be mounted upon tripod, bar or column, according to the work. The drill-cylinder, fitted to slide in the shell, is moved forward or backward by the feed-screw. The cylinder is as simple as can be imagined; a straight bore, having at each end a large opening, and a boss to which the hose is attached. The piston also is plain, much shortened in the body, with a large piston-rod, which has a long bearing in a sleeve-elongation of the cylinder.

Upon the truck is mounted an electric motor, geared to a horizontal shaft, with cranks on each end, which drive two

[1] *Trans.*, xxxiv., 928, 954.

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single-acting trunk-pistons making alternate strokes in vertical air-cylinders. One of these air-cylinders is connected by the hose to one end of the drill-cylinder and the other end of the drill-cylinder is connected by the other hose to the other air-cylinder. The air, therefore, in either air-cylinder, in its hose and in the end of the drill-cylinder to which it is connected, remains there constantly, playing back and forth through the hose according to the movements of the parts, being never discharged, and only replenished from time to time to make up for leakage. The propriety of calling the apparatus a pulsator instead of a compressor is evident.

The essential details of the cycle of operation will be easily understood. We may assume, to begin with, that the entire system is filled with air at a pressure of 30 or 35 lb. This pressure being alike upon both sides of the drill-piston, it will have no tendency to move in either direction. If, now, the motor, instead of being at rest, is assumed to be in motion, one pulsator-piston will be rising in its cylinder and the other piston will be descending in its cylinder; and, as a consequence, the pressure upon one side of the drill-piston will be increased and the pressure upon the other side will be proportionately reduced, this difference of pressure causing the drill-piston to move and make its stroke. Just before the end of this stroke, the movement of the pulsator-pistons is reversed, and the preponderance of pressure is transferred to the other side of the piston, causing a stroke in the other direction—and so on continuously. The drill thus makes a double stroke, or at least receives a double impulse, for each revolution of the pulsator crank-shaft.

Having thus sketched the general principle of operation, I will proceed to discuss some of the details. The drill-cylinder, shown in Fig. 2, while generally similar to that of the air- or steam-drill, is in many respects quite different; and especially is it remarkable for its simplicity. The usual operating-valve-chest; the valve and the complicated means for operating it; the main air-ports and the intricate little passages in and connected with the chest—are all absent, and nothing takes their place. The cylinder-heads are both solid and both fastened securely in place. The split front-head, the yielding fastenings for both heads, the buffers, the springs, the side-rods, etc.,

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of other drills, have all been banished. The cylinder is absolutely plain, with direct openings into the interior, and a boss at each end to which the hose is attached.

The piston also has been simplified. The device for securing rotation is necessarily retained; but the enlargement at the end of the piston-rod, which constituted the chuck and necessi-

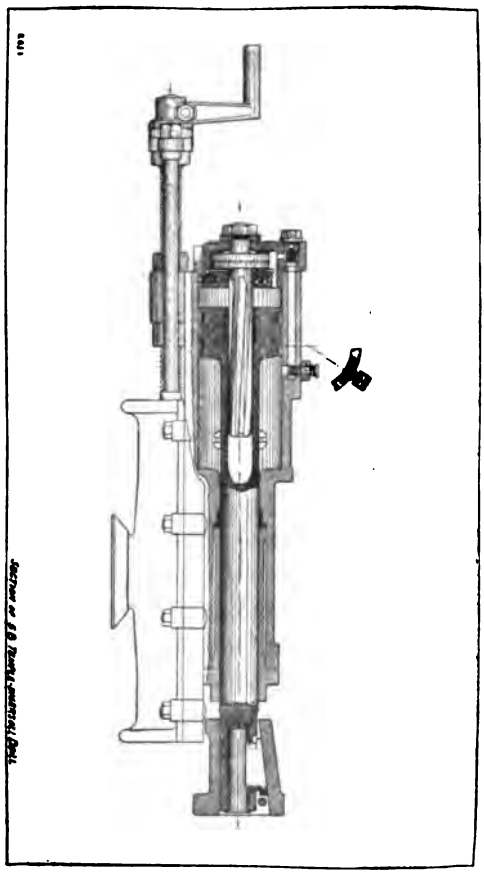

FIG. 2.—SECTION OF DRILL.

tated the split front-head, has been discarded. The piston-rod is much enlarged throughout, and a simple but effective self-tightening chuck is slipped upon the end of it.

The compressor- or pulsator-cylinders are likewise simple. There are no valves for either inlet or discharge, and there is neither jacketing nor the slightest need of it. The heating of

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the air by the compression-stroke is compensated by the cooling which attends the re-expansion of the same air, so that it does not become increasingly hot and heat the parts of the machine with which it comes in contact.

While this apparatus, as a whole, may appear complicated at first glance, it is really a great advance in simplification. The parts which it eliminates are exactly those which have always been most troublesome and expensive to maintain, and both the drill and the compressor or pulsator are the simplest ever built.

There are some minor details of this apparatus, with which it is not necessary to burden this paper, and which would involve tedious explanations not easily understood without elaborate drawings or models. In the foregoing description of the principle of operation I assumed a mean air-pressure of about 30 lb. in the apparatus. It may be asked, how this pressure is secured and maintained. When the pulsator is in operation, the air-pressure in the cylinders alternately rises above and falls considerably below the mean. At a certain point, indeed, it is below that of the atmosphere; and at this point a little valve is provided, which admits more or less air, until a sufficiency has been provided. At the beginning of operation the influx of air is rapid, so that no time is lost in getting sufficient pressure to begin with. The admission of air and also the apportionment of relative volumes thereof to the two ends of the drill-cylinder are easily adjusted by the operator.

The Electric-Air drill is not troubled by the freezing-up or choking of the exhaust, because there is no exhaust. Moreover, the air does not accumulate moisture, and the temperature does not fall to the freezing-point. Again, air becomes and remains a constant vehicle for the conveyance and distribution of the lubricant. A certain amount of oil being contributed to the system at regular intervals, it would be more difficult to prevent than to insure its reaching every working-part.

The length of hose employed seems to be limited to about 8 ft. on each side. The hose may be attached to either side of the drill, but each always to its own end of the cylinder. This length of hose gives all necessary liberty for the location of the pulsator-truck near the drill. The truck (of steel, with flanged

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wheels) is usually made for the standard 18-in. mine-track, but may be made for any other gauge. Special care in leveling is not necessary, since the pulsator will work at any angle at which the truck can stand.

Either a direct- or an alternating-current motor may be employed, the latter being preferred because it is a smaller, lighter, mechanically simpler, hardier machine, and more nearly "fool-proof." Four different speeds may be obtained with the direct-current, and two with the alternating-current motor—in the latter case, full speed for steady running and a considerably lower speed for starting a hole or working through bad ground, with immediate transition from the one speed to the other, as required. The controller is on the top of the motor and the operator at the drill can start, speed or stop the motor by simply pulling a cord, this being the only connection. The electrical connection ends at the motor; both the hose and the cord insulate the drill; and the operator is never exposed to the current.

The 5–C Electric-Air drill may be regarded as the full equivalent of the 3.25-in. standard air-drill of any make; of its comparative efficiency something will be said later. The power-requirement for this drill is from 18 to 20 amperes at 220 volts, or from 9 to 10 amperes at 440 volts—the electrical equivalent of about 5 h.p. The system being a closed circuit, this is independent of conditions of altitude, which make so much difference with the work of the air-compressor which supplies the ordinary air-drill.

The 4-C Electric-Air drill uses a 3 h.p. motor, and is a much lighter drill throughout, equivalent to a 2.75-in. standard air-drill. Table I. gives particulars of size, weight, etc., of both of these drills:

TABLE I.—*Dimensions, Weights, Etc., of Temple-Ingersoll Electric-Air Drills.*

	5-C.	4-C.
Diameter of drill-cylinder,	5⅛ in.	4.75 in.
Length of stroke,	8 in.	7 in.
Length of drill—end of crank to end of piston,	45 in.	42 in.
Depth of hole drilled without change of bit,	24 in.	20 in.
Depth of vertical holes machine will drill easily,	16 ft.	8 ft.
Diameter of holes drilled,	from 1.75 to 2.75	1 to 1.5 in.
Strokes per minute,	425	460
Horse power (at motor),	5	3

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Weights.

	Lb.	Lb.
Drill (unmounted, with wrenches and fittings),	300	192
Pulsator complete with direct-current motor, mounted on truck,	883	585
Pulsator alone,	271	160
Truck,	102	100
Motor,	400	275
Motor without armature,	330	216
Armature alone,	82	59
Controller, switch and rheostat,	72	50
Entire equipment ready for shipment, including drill, pulsator, direct-current motor, fittings, wrenches and extra parts, but no mountings, steels or blacksmith tools,	1,680	902
Pulsator complete with 30- or 60-cycle alternating-current motor mounted on truck,	630	360
Pulsator alone,	271	160
Truck,	102	100
Motor,	202	137
Motor alone,	46	34
Controller-switch with base,	45	45
Truck cross-bars for motor,	15	15
Entire equipment ready for shipment, including drill, pulsator, 30- or 60-cycle alternating-current motor, fittings, wrenches and extra parts, but no mountings, steels or blacksmith tools,	1,080	680
Tripod with weights,	540	430

NOTE.—Weight of column and shaft-bar mountings will vary with their length and diameter.

The dimensions and weights of the "Baby" or 3–C Electric-Air drill cannot as yet be put on permanent record. This drill takes the place and does the work of the "Baby" air-drill.

The Electric-Air drill strikes a blow, normally so much harder than that of the air-drill of the same capacity, that it has been found advisable in many cases in "dressing" the steel bits to make them blunter or thicker, in order to avoid breakage. The practical force of the drill had not been computed beforehand, but was demonstrated in extensive practice and experiment, and the clear and sufficient explanation came later.

The drill-piston, when running at full speed, and making a stroke for each rotation of the pulsator crank-shaft, does not strike either head. The hole by which the air enters the cylinder from the hose is located, not at the extreme end, or close to the head, of the cylinder, but a certain distance away, so that when the piston approaches the head a portion of inclosed air acts as a cushion, which first checks the piston and then

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shoots it back. The piston thus starts upon its working-stroke impelled by a certain amount of force which, we may say, has been saved over from the preceding stroke to be utilized for this. The piston after being thus started is driven forward by an air-pressure which increases as it advances, the pulsator-piston being in the attitude of chasing and gaining upon the drill-piston for a considerable portion of the stroke, while in the case of the ordinary drill-piston, driven by a constant flow of air from which it runs away, the pressure must constantly diminish as the piston-speed is accelerated. In the same way by the action of the other pulsator-piston the opposing pressure upon the advancing side of the drill-piston is a diminishing pressure instead of the constant atmospheric resistance, and these combined cause a greater unbalanced difference of pressures upon the opposite sides of the drill, a more rapid acceleration of the piston-movement, and a consequent higher velocity and force at the moment of impact of the steel upon the rock.

Perhaps the most gratifying, and also surprising, revelation of all in connection with the Electric-Air drill is the now indisputable fact that it takes only from one-third to one-fourth of the power, at the power-house, to drive it to do the same work. This is accounted for by the fact that the same air is used over and over, and that all of its elastic force is availed of in both directions instead of exhausting the charge for each stroke at full pressure. There are also no large clearance-spaces to fill anew at each stroke, as these spaces are never emptied.

A valuable feature of the Electric-Air drill, is the ability to yank the bit free if stuck in a hole and immediately continue its work. When the bit of the ordinary air- or steam-drill sticks in the hole, the drill stops and the drill-runner must free the bit as best he can. Ordinarily the feed is run up and down, the drill is hammered and things are coaxed in various ways until the bit is free. When the bit of the Electric-Air drill sticks, the motor and the pulsator-pistons do not stop. If the drill-piston is making, say, 400 strokes a minute, as soon as the bit becomes stuck the piston will receive per minute 400 alternate thrusts and pulls with full force, and nothing could be more effective for freeing the bit than these alternate thrusts and pulls. Often when the bit sticks and before the runner

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can get ready to do anything about it, the drill frees itself and is running again as if nothing had happened.

The coming of the Electric-Air drill suggests many possibilities and ominously means much to established interests. It necessarily suggests a revolution in methods and sometimes perhaps a superseding of the old plants throughout. In the working of the new drill the old central air-compressor plants are absolutely worthless, but it is not easy to imagine any general abandonment of them. After all, the result may probably be that the new drill will not to any great extent drive out the old, but will make a new field of employment for itself, and in that way lead as usual to a considerable enlargement of the already extensive business which is behind it. As has been shown, the Electric-Air drill is far from an electric drill, but the ordinary electric current, now nearly everywhere available, can be used for operating it.

In planning new installations the Electric-Air drill is to be most seriously considered. The relative final cost of operating this or any other drill, is, after all, the decisive question, due recognition, of course, being given to the peculiarities of each drill, favorable or otherwise, which are not computable, but which still have weight in determining the selection, "other things being equal."

When the Electric-Air drill is operated without its own generating-plant, the current being taken from a large power-company, some very low figures are already on record. At Idaho Springs, Colo., a mine-shaft was put down 67 ft. in 24 shifts and the total power-cost was $24 for the entire work.

In making rock-excavations for building-purposes in New York City and elsewhere, steam-drills, having a temporary boiler-installation, are frequently used. The Electric-Air drill not only avoids the expense of the boiler-equipment but will do the work at a much lower cost, the current being supplied by one of the big electric power-companies.

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The Panoramic Camera Applied to Photo-Topographic Work.*

BY CHARLES WILL WRIGHT, WASHINGTON, D. C.

(Toronto Meeting, July, 1907.)

I. INTRODUCTION.

THE application of the camera as an adjunct to topographic mapping began practically with its invention, and it has been employed with varying success since that time. With the exception of the camera to be described, the plate-camera has been universally used in this work, thus giving a projection of the area photographed on a flat surface. From such projections or photographs, by the rules of geometry and of perspective, points defining topographic features and seen from at least two camera-stations may be projected upon a ground-plane—the map.[1]

In 1904 I employed the plate-camera for this purpose in Alaska, but found that the labor necessary to plot the maps, even in a general way, was long and tedious. In 1905 a small Eastman panoramic camera was fitted with spirit-levels, a sight-alidade and a transparent scale, introduced inside the camera to register the degree-points in the sky-line of the film-negative; also, arrows to indicate the horizon-line. This was, on the whole, successful; but the details in the topography were not brought out in the views with sufficient clearness, and the photographs were too small. These difficulties were overcome by obtaining a larger and more carefully constructed instrument, which was satisfactorily employed in the field during the year 1906.

A panoramic view is made up of an integral number of perspective views upon flat surfaces, and logically is the most accurate and direct means to obtain an impression of a field of

* Published by permission of the Director of the U. S. Geological Survey.

[1] Stanley, H. M. Photographic and Co-ordinate Surveying, *Trans.*, xx., 740 to 766 (1891). Also, Flemer, J. A. Phototopographic Methods and Instruments, *Report U. S. Coast and Geodetic Survey*, pt. 2, app. 10, pp. 619 to 735 (1897).

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view for purposes of mapping. In the perspective view, only one line—the vertical center-line—records the direct azimuth or bearing of points within its perspective; and all other points must be determined by geometric projection, thus introducing an element of error. In the panoramic view, where the negative is everywhere equidistant from the camera lens and in its focal plane, the positions of all points are in direct horizontal angular relation to one another on the negative, as in nature; and by introducing a degree-scale which, at the time of exposure, is photographed in the sky-line of the negative, the bearing of any point relative to any other point within the view may be read from it directly.

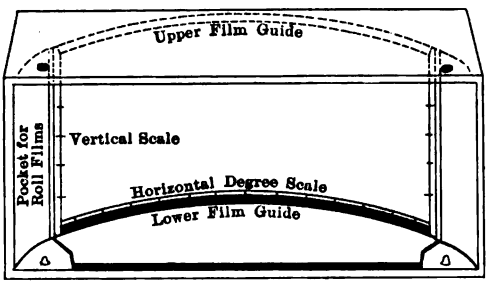

FIG. 1.—SKETCH OF PANORAMIC CAMERA WITH FRONT REMOVED, SHOWING POSITIONS OF HORIZONTAL AND VERTICAL SCALES.

The underlying principles of this method, therefore, do not differ essentially from those used in plane-table surveying. In all cases it is necessary either to have two points, within the area to be mapped, between which the horizontal distance has been determined; or, better, to have a preliminary triangulation of a number of the prominent points or peaks within the area which will form the base for the map, so that these determined triangulation-points may form tie-points for the camera-stations. It is then possible within certain areas to do the greater portion of the mapping with the aid of the camera alone. As in plane-table work, the traversing of trails and wagon-roads, and the determination of topographic stations, are more accurately made by an instrumental survey.

II. THE CAMERA.

The camera best adapted for this purpose at present on the market is the Al Vista camera, Model 5 B, which takes a 5-

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by 12-in. view, including an angle of 140° (Fig. 2). It consists of an oblong box, 6 in. by 6 in. by 11 in., fitted with a lens of fixed focus, which may be made to revolve at different speeds within a half-circle, and with the introduction of apertures of various sizes allows the correct light-value for the exposure to be obtained. Retaining the film are two circular film-guides, so placed, one at the top and one at the bottom of the camera-box, that the film, in passing outside of them, will be in the focal plane of the lens at all points. For more exact work, a camera of this type should be constructed of metal, to eliminate the error which may be introduced by shrinkage of the wooden parts.

To fit the camera for surveying, both a horizontal and a vertical scale should be so adjusted within the camera-box as to be photographed on the sensitized film at the time of exposure. To accomplish this, a narrow strip of celluloid graduated into degrees ($1° = \frac{2\,r\,\pi}{360}$ when $r =$ focal length of lens) is glued to the lower circular film-guide in such a way that the degree-marks project $\frac{1}{8}$ in. above the guide. The sensitized film passes on the outside of this, and when the exposure is made the degrees are photographed in the sky-line of the film. These degree-marks are nearly 0.1 in. apart on the negative; and on an enlargement of two diameters, angular readings may easily be estimated to an accuracy of 5 minutes.

On the sides of the upper and lower film-guides is attached a thin metal strip, 0.25 in. wide, in which wedge-shaped notches are made, the divisions ($= \frac{1}{20}$ in. for the camera used) being equal to a hundredth part of the focal length of the lens (see Figs. 1, 3 and 4). These two vertical scales must be adjusted so that the center-point marked on each strip will fall in the center of the camera-field or film, and a straight line connecting these two points on the negative, after exposure, will thus establish the horizon-line. To the top of the camera-box two 60-second levels at right angles to each other are adjusted, and a sight alidade is attached, in such a manner that its direction of sight coincides with a line extended from the center of the lens to the zero or center-point of the horizontal scale inside the camera. The bottom of the camera-box is fitted with three leveling-screws, adjustable to the transit-tripod. To

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make the camera more serviceable, and to eliminate the necessity of a transit at the camera-stations, a transit-plate with vernier may also be fitted to the camera-box; and from this all

FIG. 2.—THE PANORAMIC CAMERA FITTED FOR TOPOGRAPHIC MAPPING.

necessary angles can be read. From the photographs taken with a panoramic camera fitted with the above attachments it is possible to determine the positions, both horizontally and vertically, of all points within the area photographed.

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FIG. 3.—RENDU GLACIER, GLACIER BAY, ALASKA.

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FIG. 4.—RENDU GLACIER, GLACIER BAY, ALASKA.

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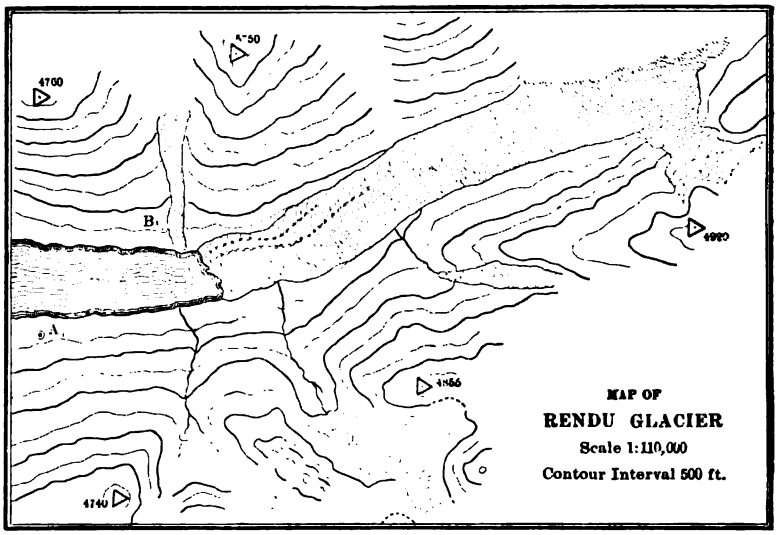

FIG. 5.—MAP OF RENDU GLACIER, GLACIER BAY, ALASKA. PLOTTED FROM FIGS. 3 AND 4.

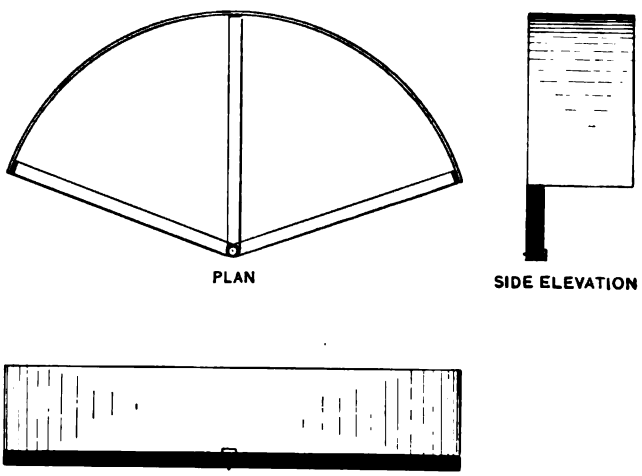

FIG. 6.—SKETCH OF FRAMEWORK FOR PLOTTING DIRECTLY FROM PHOTOGRAPH.

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III. Adjustment.

To determine whether the zero of the horizontal scale corresponds to the point of sight taken with the alidade, and also to ascertain whether the center-marks of the vertical scale representing the horizon-line are correctly placed, the procedure is as follows:

With the transit set up over a station from which a broad view may be obtained, sight on some prominent point and then take angular readings to other points and note several points of equal elevation to the station occupied. Set up the camera over this station, sight the attached alidade on the first point sighted with transit, and then take view. After development of the negative a comparison of the readings is made. If the zero of the horizontal scale does not fall directly over the point sighted with the alidade, shift the scale or alidade to the amount of difference indicated on the negative. If the points of equal elevation noted by the transit-readings do not fall on the horizon-line represented on the vertical scales, it is a simple matter to adjust either the scales or the levels until such is the case. To make this latter adjustment perfect may require several trials.

A simpler way to adjust the vertical scales, and a test that should be made from time to time in the field, is to set up the camera on the shore-line of a lake or at sea-level and make an exposure; the horizon-line indicated by the vertical scales should coincide with the surrounding shore-line of the lake or the horizon-line of the sea. If this is not the case, the scales should be adjusted until harmony exists.

With the present camera, adjustments for determining the parallelism of the film to the axis on which the lens revolves cannot be made. A camera, however, is under construction in which this and other adjustments will be possible.

IV. Field-Work.

The accuracy and detail of the mapping depend largely upon the precision of the base-work, the number of camera-stations occupied, and also the scale of the map. If a careful primary and secondary triangulation of an area has been made, and

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numerous stations have been established, one could enter the field with camera alone and complete the field-work of a fairly accurate topographic survey of that section in the same manner one would use a plane-table, but in a much shorter time. With the plane-table, most of the plotting is done in the field, with the camera the same work is done in the office; with the plane-table one can occupy but one station at a time; with the camera, when plotting, one can practically occupy two or more stations at once and in this way simultaneously, from two or more points of view, see the area to be mapped. This latter circumstance is a strong factor in the identification of points. Again, when a mere reconnaissance-survey is to be made, the topographic work may be accomplished with the camera while the triangulation is advancing. By so doing, not only time is saved, but much information regarding the timber-limits and density of growth, and in some instances characteristics in land-sculpture and geologic structure, may afterwards be obtained from the photographs, all of which would be omitted by the usual method of mapping. For reconnaissance-work the camera is, therefore, especially to be recommended.

All the triangulation-stations occupied by the transit should also be occupied by the camera, thus securing a check on the triangulation. In selecting camera-stations, those taken at low and moderate elevations are of greatest service as aids in drawing the topographic details, while those at high elevations are best for advancing the triangulation. In each case, however, it is necessary to have three previously-determined points in the field of view, preferably at wide angles, which may be sighted with the alidade on the camera from the station occupied, so that, from the angular readings taken, the position of the camera-station can be determined. The elevations of these intermediate stations are noted from barometric readings and later are checked by determinations from the elevations of known points included upon the negative. The elevations of the triangulated points, where the transit is employed, are determined in the usual way by vertical angle-readings upon points of known elevation or at sea-level.

It is hardly necessary to mention the advisability of erecting monuments or signals upon all stations so that they may be sighted from other and new stations.

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V. Plotting Maps from Photographs.

It is first necessary to develop the film-negatives—the best plan being to do so in the field, so that a defective negative may be replaced by taking another view at once. Such failures, however, are not probable. During the past summer I took 200 views under all conditions of weather, and not one of the negatives was defective. They were developed in the field, for the most part within a day or two after exposure; and several dozen, left undeveloped until my return to the office, some three months later, were developed with equally good results.

A preliminary plotting of triangulation-points and camera-stations on the map should be done in the field, and traverses of trial- and shore-lines should be added. This is recommended to prevent the introduction of "holes" in the map, that is, small areas which cannot be accurately plotted in the office, because they were sighted from but one station, or were otherwise neglected. In the office, to facilitate the topographic plotting, enlargements are made from the negatives, which were 5 by 12 in., to prints, 10 by 24 in. in size. As the horizontal and vertical scales are photographed on the negative at the time of exposure, no error is introduced by making these enlargements. The uneven contraction and expansion of photographic paper is often a cause of error in plotting; but if care is taken in developing and drying the prints, and if they are all made on the same kind of photographic paper, this error is only a slight one. To eliminate it, however, plotting may be done directly from the negatives.

Having established a map-scale and plotted the triangulation-points, the camera-stations should next be determined. This is done by the three-point method, by which the angular readings to three or more determined points, taken from the station, either by a transit or read from the photograph, are plotted on a piece of tracing-paper. The lines indicating the angular readings are placed over the respective points and the positions of the camera-stations thus located.

The camera-stations being plotted on the map, one may next proceed to select numerous points common to two or more views from different stations, and to plot their positions by lines of intersection representing their respective bearings read from the horizontal scale on the photograph.

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To facilitate this work a T-square 1 ft. long is made, the vertical arm being divided to correspond with the vertical scale of the photograph. The photograph is adjusted to a drawing-board 12 by 24 in. in size, so that the center-line of the vertical scales coincides with the center-division marked on the T-square. (See Fig. 7.) On the map a line is drawn from the respective camera-station in the direction sighted at the time the photograph was taken; and by the aid of a protractor, and angular readings taken with the T-square from the horizontal scale, the directions of all points may be plotted. When the position of any point has been determined by the intersection of its lines of direction from two or more stations, its relative elevation may be obtained by multiplying the number of divisions on the T-square above or below the center-line by the horizontal distance of the point, measured in feet, and dividing by 100, as the divisions represent hundredths of the focal length for the photograph. (The focal length or radius for the enlarged panoramic photograph is obtained from the formula $r = \frac{2}{\pi} H$, where $H = \frac{1}{4}$ of the circumference of circle or distance measured between 90° on the horizontal scale of the enlarged photograph). This result is added to the elevation of the camera-station above sea-level and the sum is the elevation of the point. Thus, if a mountain-peak is 20 divisions above the center-line on the vertical scale and the horizontal distance measures 8,000 ft., then its relative altitude is $\frac{20 \times 8{,}000}{100} = 1{,}600$ ft.; this added to the elevation of the camera-station gives the total elevation of the peak above sea-level. (In computing the elevation of distant points the correction for curvature of the earth and refraction should be added.)

When plotting, the photographs showing the same area from two stations are set up on the drawing-table, as shown in Fig. 7, and points seen from both stations are numbered directly on the photographs, the same numbers being used to indicate the points on the map.

The protractor used in this work was of card-board, 10 in. in diameter, with the central portion cut out. This was oriented on the map over the camera-station and made fast with thumb-tacks. A narrow scale, 10 in. long, divided into

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units of the map-scale, and fitted with a pin-point at the 0 mark, served as a protractor-arm as well as a scale for measuring the horizontal distance. This arrangement is shown in Fig. 7.

To obviate the necessity of orienting the protractor for each station, and to facilitate plotting, a parallel movable engineers' protractor may be used to advantage. As this protractor has a fixed orientation it is necessary to determine the azimuth for the zero-point on each photograph and to adjust the horizontal scale so that the direct azimuth readings may be taken from the photographs.

After a sufficient number of characteristic points have been plotted upon the map and their elevations noted, it is a simple procedure to introduce the lines of contour by the aid of the photographs. By using the above method an experienced to-

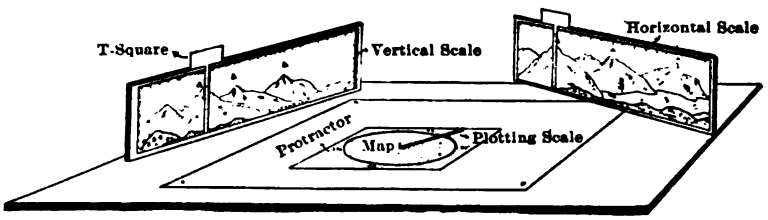

FIG. 7.—SKETCH SHOWING ARRANGEMENT FOR PLOTTING OF MAP FROM PHOTOGRAPHS.

pographic draftsman can complete an expressive map of the land-sculpture of an area from photographs in the office.

Both the plotting of points and the determination of their elevation may be done graphically as follows:

Construct a framework, consisting of 3 bars, equal in length (about 5 in.), attached to a curved strip of metal sheeting, of the size of the photograph, shown in Fig. 6. At the junction of the 3 bars representing the radii of the arc, attach a pin-point; the distance from pin-point to outer surface of curved metal sheet should be equal to the determined focal length for the photographs from which the map is to be plotted. On the photograph, project the points to be plotted to its base (bottom of photograph). Fasten this photograph face out to the framework and place pin-point on the same over the station-point on the map and then orient it in the photo direction. Project the points at base of photograph on the map, indicating them by numbers corresponding to those on the pho-

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tograph. In this manner the direction of each point is plotted directly from the photographs.

To ascertain the altitude of points on the photograph relative to the camera-station we have this proportion: The measured distance, y, of point on photograph above or below the horizontal line is to the determined focal length for photograph, f, as the elevation of the point E above or below camera-station is to its horizontal distance, x, from the camera-station. This proportion, $y : f = E : x$, gives $xy = Ef$, which is the formula for a hyperbola in which x and y are the variable and E and f the constant factors.

To construct the diagram, divide the horizontal axis of a rectangular co-ordinate-system into units of the map-scale;

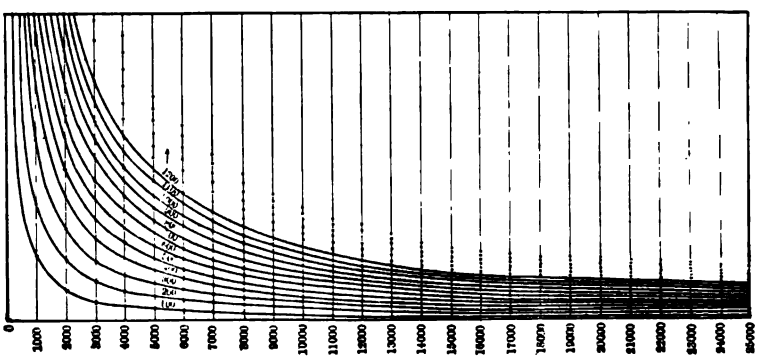

FIG. 8.—DIAGRAM FOR DETERMINING ELEVATION OF POINTS.

from this, at points representing intervals of 1,000 ft. horizontal distance, erect vertical lines, on which the computed lengths of y for each 100 ft. elevation at the respective horizontal distances are indicated. By connecting the points which represent the same elevation on these vertical lines of the diagram, hyperbolic curves are formed (see Fig. 8), each curve representing points of equal elevation at their respective horizontal distances. With a diagram thus constructed, we can scale the horizontal distance of a point directly and also measure its elevation, relative to the camera-station, by transferring, preferably with a pair of dividers, the distance y from the photograph to the diagram, directly above the point indicating its horizontal distance. The practicability of this graphic method has not been tested; but, though it may lack in accuracy, it

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will undoubtedly facilitate and greatly shorten the work of plotting the maps.

VI. Accuracy of Method.

The two photographs, Figs. 3 and 4, may serve to illustrate the panoramic method. These views are of Rendu glacier, in Glacier bay, Alaska. The peaks with a triangle over them are triangulation-points determined in 1892 by Professor H. F. Reid, whose map was used as a base. The two camera-stations occupied, on opposite sides of the inlet, were located by transit-readings on the triangulation-points. These points and stations were first plotted on the accompanying map, Fig. 5; the positions of numerous points represented on each photograph were determined by the above method, and a map sufficiently accurate for the intended purpose was thus constructed.

The maximum difference in elevation of a point measured on photographs from two or more camera-stations seldom exceeds 10 ft. for a horizontal distance of one mile. Determinations of the two triangulation-points shown in the accompanying views gave the following results:

Triangulation-Pt. 1, from Sta. A. Horizontal dist. = 31,500 ft. Elevation = 4,934 ft.

Triangulation-Pt. 1, from Sta. B. Horizontal dist. = 25,000 ft. Elevation = 4,906 ft.

A difference of 28 ft. in a distance of 5 miles.

Triangulation-Pt. 2, from Sta. A. Horizontal dist. = 14,500 ft. Elevation = 4,868 ft.

Triangulation-Pt. 2, from Sta. B. Horizontal dist. = 18,400 ft. Elevation = 4,842 ft.

A difference of 26 ft. in a distance of 3 miles.

VII. Advantages of the Panoramic Camera Method.

Photography cannot replace instrumental topographic surveying, and, in many areas, cannot be used at all; yet experience has shown it to be a valuable adjunct in nearly all surveys. The topographic features of an area will necessarily determine the method to be employed in its mapping. A flat, rolling plain cannot be mapped in the same manner as a rugged mountainous region, or a heavily timbered area where broad views cannot be obtained. Areas of bold topography

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with sharp or often rounded summits and little timber, such as are encountered in the western States and Alaska, are most advantageously mapped by the camera method. To make a plane-table survey of such an area requires the expenditure of much time in the field in constructing the map, and the expense of a large field-party. The conditions of weather which exist along the western coast and in Alaska are such that the summits are enveloped in a cloud of fog, often during the greater portion of the day. When the view is clear for a short time only, a photograph of the surrounding country may be taken, thus completing in a fraction of an hour the work at a station which would require many hours or even days by the plane-table method. It should be noted that alidade-reading and note-taking from a mountain-top in Alaska on the usual cold, foggy day, with a still colder wind blowing, is by no means comfortable, and even the best topographers are inclined to hurry their work. In this manner, some portions of an area are more carefully mapped than others. Having the photographs of an area with the bearings of all points registered directly above them, the plotting of the map can be done with more agreeable surroundings in the office, where all conveniences are at hand, and thus a map can be constructed of more uniform accuracy, and with as much detail as desired. If this or a similar graphic method of mapping were applied in certain regions a great deal would be saved in both time and expense, compared with topographic surveying as generally practiced.

VIII. Comparison of Panoramic Camera Method with Other Photographic Methods.

In the present practice of photo-topography a photo-theodolite or similar instrument is employed. This consists of a specially prepared plate-camera with a fixed-focus lens adjusted to a horizontal transit-circle. Attached to the top of the camera-box is a telescope with vertical circle for the reading of vertical angles. At the back of the camera and directly in front of the sensitized plate are two cross-hairs, the one forming the vertical center-line and the other the horizon-line. Inside the camera is a flat magnetic needle attached to a disk, to which a vertical transparent scale is adjusted. This revolves directly in front of the sensitized plate so that when the expo-

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sure is made the magnetic bearing of the view is photographed upon it. With this instrument a group of eight views are necessary to complete a panorama, and the angular direction of each view must be read on the horizontal transit-plate and noted. Another disadvantage of the photo-theodolite is its weight and bulk, besides the limited number of plates which may be taken on a trip and the necessity of reloading the plate-holders. With the panoramic camera the daylight-loading films are used, which eliminate the danger of breakage, lessen the weight to be carried and permit development in the field directly after exposure, in a daylight developing-tank.

The plotting of a map from the views taken by the photo-theodolite is a long and tedious process (though possessing no special difficulties), and the office-work necessary to complete the map is many times greater than that required for the same amount of mapping by the panoramic camera.

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Search for the Causes of Injury to Vegetation in an Urban Villa Near a Large Industrial Establishment.*

BY PERSIFOR FRAZER, PHILADELPHIA, PA.

(New York Meeting, April, 1907.)

INTRODUCTION.

For various reasons I have not specified the locality where the research indicated in the following pages was undertaken. It will suffice to say that it was on the grounds of a villa once remote from, but now completely surrounded by, its neighboring city, and in close proximity to an industrial establishment of great extent and importance manufacturing many kinds of steel articles and employing upwards of thirty chimney-stacks for power and process work.

The problem was to discover the cause or causes of the mortality to trees and plants on the place, and to trace these causes to their origin.

Before undertaking this experimental work, the bibliography accompanying this paper was compiled and studied.

It appears from an examination of the careful scientific work performed by the ablest French, German, and English chemists during the last 25 years that they are unanimous in assigning the principal—in fact, the overwhelmingly predominant—cause of the destruction of vegetation to sulphur oxides, resulting either from direct oxidation of sulphur in oil of vitriol works, etc., or from the oxidation of the sulphur from the minerals associated with commercial coal.

Schröder and Schertel showed in 1884 that the sulphates deposited upon the leaves are not injurious to the plants; and Freytag proved that free sulphuric acid could not be found

* SECRETARY'S NOTE.—The manuscript of this paper was received in June, 1906, and the paper was read by title at the London Meeting, July, 1906. But since there was no opportunity at that meeting to present its contents, even in oral abstract, the paper was transferred, with the author's consent, to the New York Meeting. The foregoing statement will fix its real date (so far as questions of priority are concerned) in June, 1906.

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in the soil because the atmospheric water carried away all which did not immediately combine with its basic constituents.

The sulphates which accumulate in the ground are highly favorable to plant-growth—especially lime sulphate, which is partly carried into the tissues of the plants and increases the percentage of sulphur in the ash of the leaves analyzed.

The old method of determining the cause of injury by analyzing the leaves of plants is open to the objection that the sulphuric oxide detected is due partly to a harmful and partly to a harmless source, while the amount to be ascribed to each is indeterminable.

In the investigation here described, which I pursued for six months, I was assisted by my son, Mr. John Frazer, Instructor in the Chemical Department of the University of Pennsylvania.

I concluded to follow the old established method pursued by Freytag, Schröder, Schertel, Reuss, Haselhoff and Lindau, Haywood of the Agricultural Department at Washington, and many others, which consists in analyzing the plant and the soil of the region of injury, and for comparison, those of a region remote from the injurious action, and deducting the amount of sulphuric oxide in the latter from that found in the former, to ascertain the excess due to the acid-producing works. But in addition to this method, by means of apparatus devised for the purpose, I measured the amount of sulphur oxides in the air coming from the works, and compared it with that coming from other directions. This method established the source of the injury directly instead of indirectly; yet in spite of its obvious advantages it seems never heretofore to have been employed except by Prof. Mabery in 1894. This is strange, the more so that in the fifth chapter of Haselhoff and Lindau's classical work it is stated that "another way to prove the effect of acid smoke-gases on plants is the demonstration of the smoke-gases with reference to their injurious constituents in the air of the locality of plant-injury in question." But the method thus mentioned was that employed by Braconnot and Simonin near Nancy in 1848, and consisted simply in the use of litmus-paper placed at different distances and in different directions from the place of emission of the furnace-gases. By that method it is possible to ascertain only that there are acid vapors of some kind in the

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air, but it does not prove, even qualitatively, the presence of sulphur oxides, the predominating influence of which in the destruction of vegetation was not known at that time.

It is worth while in this connection to quote the conclusions reached by Haselhoff and Lindau as the result of their own exhaustive researches and those of all of their predecessors up to 1903, which are found on page 143 of their book, at the end of the general treatment of the subject; and may be translated and epitomized as follows:

1. Even by strong and repeated additions of sulphurous and sulphuric oxide to soils no essential increase of the sulphur-content is effected; no change of the constitution of the soil takes place; and therefore injury to the plants through the soil is out of the question.

2. Direct action of free sulphurous or sulphuric oxide on the roots of plants is improbable. Should an increase of sulphates occur through the action of smoke-gases on the soil, this would have no injurious effect upon the growth of plants, and may be excluded from consideration.

3. An injurious effect on plants can only occur when the acid gases come into direct contact with the leaf-organs of plants. By injury of plants through SO_2 the content of SO_3 in the plant is always increased; but as this occurs also when the content of sulphates in the soil increases, the observation by itself cannot prove injury through acid gases. The peculiar conditions of each place must be considered.

4. The susceptibility of different plants to sulphurous and sulphuric oxides is different; and even the same plants show different degrees of susceptibility according to their location.

5. Long exposure to even so small a quantity as one millionth of sulphurous acid has been found injurious.

Schröder considers this acid less, Freytag more, injurious to vegetation than sulphuric acid.

6. The various quantities of sulphuric acid collected from the same surface of leaf of two different plants under approximately the same circumstances will not of themselves afford a measure of the injury done to the whole organism of the plants; on the contrary, the specific peculiarities of the several plants must be taken into account and submitted to proof.

7. The cracks in the leaf-organs have nothing to do with the absorption of sulphurous acid. The gas is not absorbed through these cracks but by the entire leaf-surface, and the amount of absorption depends upon the peculiar organization of the leaf.

8. The effect of the absorption of sulphurous acid is to disturb the circulation of water. This appears in an increased extrusion of water, and results in the drying of the leaves.

9. The absorption of sulphurous acid, and consequent disturbance of the circulation of water in the plant, is, for the same proportion of sulphurous acid to the air, greater for a given time with light, high temperature and dry air, than with darkness, low temperature and moist air.

The sulphurous acid and acid smoke-gases in general are more injurious by day than by night.

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10. Morphologically, the effect of the sulphurous acid is shown by the formation of spots on the leaves, the death of the leaves and twigs, the retardation of the rings of growth, and finally the destruction of the plant.

11. In the interior of the cell plasmolysis is induced; the grains of chlorophyll are destroyed, and finally form with the plasma and other materials a brown amorphous mass. At the same time, in most cases, especially if the injury has been gradual, tannin separates out, as can be detected by brown or black nodules in the cells.

12. The mode of action of the sulphurous acid is to be conceived as a disturbance of the life of the plasma in the cell. It probably acts as sulphuric acid produced through the oxidation of sulphurous acid by the oxygen of the assimilating chlorophyll grains in the presence of water from the cell-sap.

13. By the continuous action of rain or water from other sources, the sulphuric acid of the dead leaf-organs which has been taken from the air may be again eliminated. In conifers, and probably other plants, of which the organs are gummy or waxy, the sulphurous or sulphuric acid taken from smoke-gases is not further neutralized in the mass, so that the recognition of smoke-injury is impossible.

14. No absolutely sure botanical means exists of recognizing the injuries by sulphurous acid, but it is possible, only through the complex of outward and interior injuries, to conclude their presence. The surest proof is the chemical determination of sulphuric acid.

I. Journal of the Investigation.

1905.

February 24.—There were brought to me some papers and a blackened cloth relating to the case of the smoke-nuisance. I began at once a series of experiments with the object of ascertaining whether the injuries complained of were actual; if so, to what substances they were due; and what was the source of the injurious substances.

The cloth originally had been white; but by exposure for a few days to the air on the lawn it had become changed to a dark gray.

March 28.—The weather being for the first time favorable for observation, I removed the heads of three flour-barrels, and placed on an expansible hoop, in the middle of each, a diaphragm of cloth saturated with alkaline carbonates. The cloth had been previously tested and proved to contain no sulphur or sulphates, and the weight of its ash for a given area had been determined. The barrels with their cloth diaphragms were then placed at points A, B and C on the map, Fig. 1, the axes of the barrels pointing in the direction of the industrial works, which were suspected of contaminating the air. The cloths were thus exposed for about four days.

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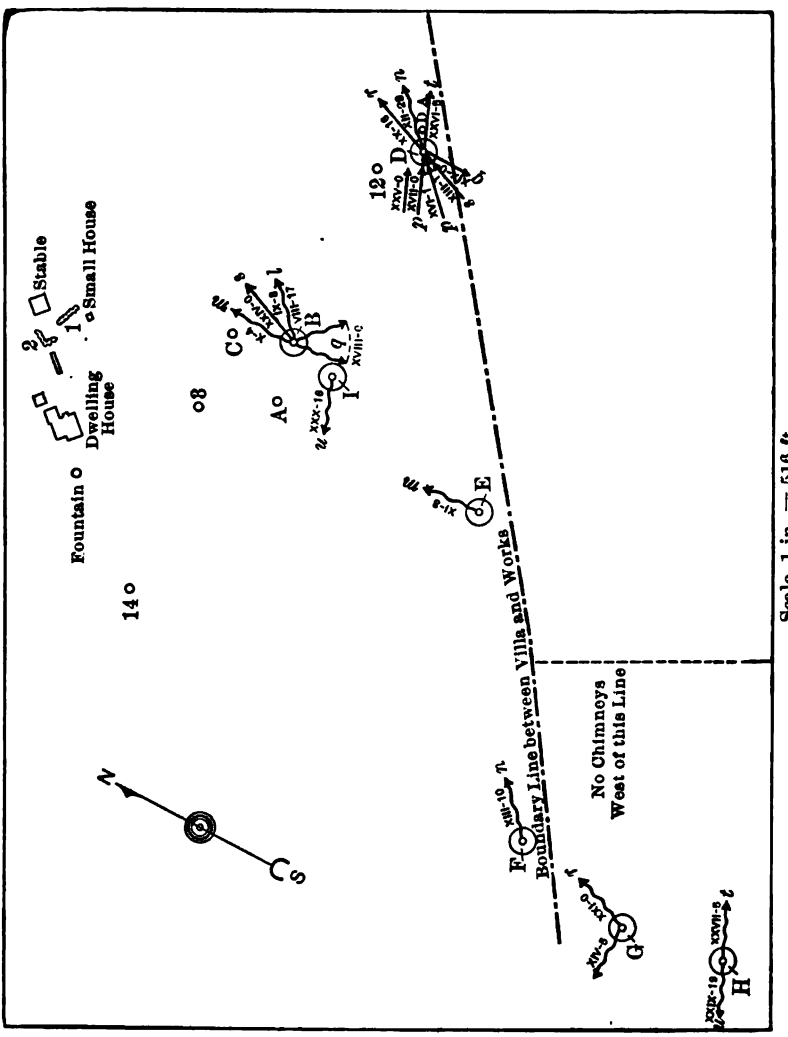
March 31.—The cloths at A, B and C, Fig. 1, were removed and replaced by others.

April 16 to 18.—I visited Pike county with Mr. John Frazer



Scale, 1 in. = 516 ft.

FIG. 1.—SKETCH-MAP, SHOWING GENERAL SITUATION AND POINTS OF OBSERVATION.

to obtain from a region free from coal-smoke specimens of vegetation similar to that under examination.

April 22.—The cloths exposed March 31 were removed and replaced by others.

Examination showed that in every case the cloths had taken up from the atmosphere a large percentage of their respective

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weights, varying from 6.06 (A, second experiment) to 10.85 (A, first experiment) per cent. of SO_3. In other words, a cloth about 20 cm. square absorbed, during four days' exposure to the air, 10.85 per cent. of its own weight of SO_3 from the atmosphere.

These experiments proved conclusively that there existed in the air, at least at intervals, during the period from March 28 to March 31, large quantities of the poison which was respon-

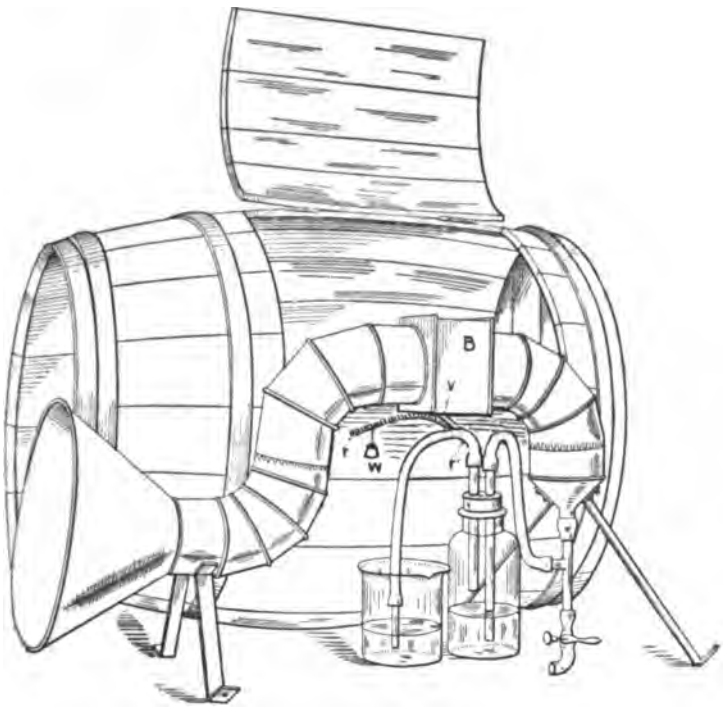

FIG. 2.—AUTOMATIC AIR-SELECTER.

B, square box uniting two systems of jointed pipes;
v, valve in the bottom of the box; held in place by
r, rod of stiff wire attached to the wooden valve, and pressing it into place by means of
w, a weight suspended from the free end of r.

sible for the injury and death of the plants. The results will be found in Table I.

The automatic apparatus for examining air coming from a given direction was set up at A, in a barrel arranged for the purpose.

Fig. 2 is a view of this apparatus. It consists of a funnel-shaped mouth, of which the angle between the sides of the

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cone-frustum and the base is such that currents of air which are not approximately in the direction of the axis of the jointed pipe immediately behind it cannot exert pressure enough to pass through the solution contained in a Wolf-necked bottle at the posterior extremity of the conducting-pipe. Consequently, by setting the apparatus in such a position that lines parallel to the axis of the barrel passed through some part of the works suspected of emitting the noxious gases, it was rendered certain that no air from other directions could enter the solution.

The jointed pipe was curved upwards and again downwards to prevent rain from entering the fixing-flask. It was furnished with a square box in the middle or highest part of the curve, in which was a hinged safety-valve connected with a wire lever, to prevent a too-strong wind from scattering the solution and breaking the apparatus. A very slight weight was sufficient to keep the valve closed while the air passed through the solution.

From the second tube of the Wolf flask a glass tube connected by a rubber tube dipped into a beaker containing a little water, to seal the fixing-flask from the air.

The whole apparatus was secured by iron bands and clamps screwed to the inner sides of an ordinary flour-barrel, into the upper half of which a door had been cut and hinged for the purpose of permitting free manipulation.

A solution of alkaline carbonate was placed in the Wolf flask, which, when the connections were made, was left for a varying number of days to receive the sulphur oxides carried by the wind from the direction of the works.

The records of the results of these experiments will be found in Table I. As the latter were qualitative, they were substituted later by quantitative experiments in which a known volume of air from the pipe was pumped through the solution.

April 24.—The automatic air-selecter was moved from A to B, and the air-valve was closed by means of a light weight.

May 1.—The contents of the fixing-flask at B were removed and the bottle was refilled with alkaline solution. The solution removed contained 0.0125 g. of SO_3. For future quantitative measurements, a small force-pump was calibrated.

May 5, 3.30 p.m.—Wind steady from the east. Passed 10,000 cc. through the solution, taking the air through the automatic

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apparatus at B. The direction of the wind was not favorable for bringing products of combustion of the works into the apparatus. The solution exposed in the air-selecting apparatus since May 1 contained 0.0047 g. of SO_3. The air examined on the spot contained 0.42 g. SO_3 per cu. meter.

May 16, 3.15 p.m.—Wind squally, about SW. Atmosphere damp and threatening rain, which later descended in torrents.

Points A, B and C were established by survey and the contents of the fixing-flask were taken for analysis.

May 29.—Wind E. of S. and day cloudy. No visible smoke passed over the lawn but an odor was noticeable. At 4.25 p.m., the wind had shifted to W. Passed 10,000 cc. through solution VII.

During the succeeding fortnight plans were put together and a tracing made on vellum. The points occupied as stations of observation were platted on this map.

June 6.—Temperature 92°. The weather was very oppressive. Wind variable and gusty from SW. to W. Heavy rain fell as a test was about to be made.

June 9.—Temperature 80°. Wind variable, but generally SW. and W. No smoke visible over lawn. 10,000 cc. of air were passed through a half-saturated solution of sodium carbonate, and called VIII.

A second test was made by passing 10,000 cc. of air through a $\frac{2}{5}$ saturated solution, called IX.

June 15.—Wind SE., but shifted to SW. and W.

Two $\frac{1}{3}$ saturated solutions were prepared, through each of which, X. at station DA, and XI. at E, were passed 10,000 cc. of air.

While making the last collection the wind changed again and heavy volumes of smoke were poured over the site of the station first occupied.

June 17.—Wind SW., variable and puffy. Sky cloudy. 10,000 cc. of air were passed through solution XII. at station D, and an equal quantity through XIII., Station F.

June 20.—Wind variable from NE. and E. With Mr. John Frazer, occupied a point to the west, G. The smoke rose high but its odor was noticeable. 10,000 cc. of air were passed through solution XIV. Afterwards a similar volume of air was passed through solution XV. at E.

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Collections and determinations XVI. to XXIV., inclusive, will be found in Table III., p. 514.

Sept. 12, 8.30 a.m.—Wind SW. to W., but variable. Passed 10,000 cc. of air through XXV. at DA.

In spite of the heavy rain which had fallen during the entire previous day and night, the branches and leaves were covered with black smut, which soiled any object touching it.

Specimens of the withered leaves of an American beech-tree were gathered for analysis 12.

An old and a younger hemlock hedge had lately been trimmed by the gardener and the sickly parts removed. Specimens of each and of a Norway spruce were taken for analyses 1, 2 and 3.

A visit was made to a country-place about $\frac{1}{8}$ of a mile to the north. Here was a hemlock hedge planted 32 years ago. The gardener stated that the vegetation shows no injury from smoke, and no unusual amount of smoke is noticeable, except from a small saw-mill in the neighborhood which burns soft coal with its shavings.

Sept. 16, Noon.—Wind W. to SW., variable. Took specimens of the subjects of analyses 1, 2, 3, 4, 7, 12, 13, 14. Occupied a new station at H to the west, where 10,000 cc. of air were passed through solution XXVII. No odor of smoke.

Sept. 26.—Eight porcelain crucible-covers were weighed, coated inside with vaseline, and re-weighed. These were placed at various carefully noted points on the place, and left to collect the soot, care being taken in the selection of points to avoid other contamination. Cloths saturated in litmus solution were also suspended at various locations.

Specimens of soil were taken from the villa grounds, from the place to the north previously mentioned, and from "Airdlie," and Silver Lake in Pike county, Pa., for analysis and comparison as to sulphuric oxide content.

Oct. 7.—Wind N-by-E. Temperature 42°. Clear.

The porcelain covers left eleven days ago were collected. Several of these were, from various causes, not able to furnish reliable tests, and were rejected.

A new and larger pump was arranged for future experiments.

Oct. 14.—Visited works near those suspected of contaminating the air, to ascertain how much of the smoke-injury could be ascribed to the former. The works consume about 8,000

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tons of bituminous coal per year, and have five smoke-stacks, from which, at the time of the visit, little or no smoke was seen to issue. Two Hawley down-draft furnaces were in use. Mechanical stokers had been used, but subsequently abandoned.

Oct. 21.—Subjects of analyses 1, 2, 3, 12 and 14 were collected, as well as a specimen of soil from near the hemlock hedges, for analysis. Observations of the smoke were made from a hill opposite the works.

Oct. 27.—At 9.21 a.m. a site was selected and located by compass and measurement from which to observe the smoke. This site is lettered "I" on the map.

Wind NE. to E., variable. Occupied station H and passed two measured volumes of air, each through a separate solution.

The first experiment failed owing to a defect in the new pump, tried for the first time. The second experiment, with the old pump, was successful. Passed 10,000 cc. through solution numbered XXIX. Odor of smoke occasionally noticeable.

Occupied station I and passed 10,000 cc. air through a solution marked XXX.

From 10 to 11 a.m. the wind varied from a trifle W. of N. to NE., and finally at noon it was nearly E., carrying much smoke over the lawn. At 4 p.m. the wind was very light. The columns of smoke and steam were nearly vertical. Black masses of smoke poured out of the ventilators of one of the buildings and also from the stacks behind this building and from the boiler-house to the southeast.

II. Proof by Qualitative Determination of Sulphur Oxides in the Atmosphere.

The experiments with the saturated cloths, of which the results are condensed in Table I., furnished a striking and conclusive proof of the amounts of sulphur dioxide and trioxide (SO_2, SO_3) with which the air was charged at intervals. Pieces of cloth, 27.94 by 16.14 cm. in area, and weighing each 1.896 g., collected from the air in four days amounts of SO_3 varying from 6.8 to 10.85 per cent. of this weight. Similar cloths exposed for 22 days showed a smaller content of this gas; which was due to the fact that a portion of the fluids with which they had been saturated during the prevalence of damp weather, dripped from them to the under side of the containing barrel.

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It was not established by these experiments at what period the maximum was reached, but only that the air contained at intervals quantities of sulphur oxides much more than enough to effect the observed destruction of plants.

Roman numerals are employed for the analyses establishing the existence of sulphur oxides in the atmosphere. Those from I. to VI., inclusive, are applied to the qualitative, and VII. to XXXI., inclusive, to the quantitative determinations. At A, analyses I. and II. were made; at B, analyses III. and IV. (IV., however, was lost); and at C, analyses V. and VI.

Analyses I., II., IV. and VI. were to determine the sulphur trioxide content of the fixing-bottle in the automatic air-selecter.

I. Collection from April 24 to May 1.

II. Collection from May 1 to May 5.

IV. Collection from May 5 to May 16.

VI. Collection from May 16 to May 29.

All of these showed qualitatively the presence of SO_3, which must have come from the direction of the industrial works.

Nos. III. and V. were experiments with measured quantities of air forced through the solution by the pump.

No. III. was rejected, owing to doubt of the correct adjustment of the apparatus. No. V. was lost during analysis.

TABLE I.—*Proof of the Existence of Sulphur Oxides in the Air, from Experiments with Saturated Cloths Free from Sulphur.*

Station Where Exposed.	Period of Exposure, 1905.	Distance and Direction from the Northwest Corner of the Western-most Building of the Works.	Per Cent. of Weight of Cloth.		Remarks.
			Ash.	SO_3.	
A	March 28 to 31.	731 ft. W. 37° N.	1.44	10.85	
A	March 31 to Apr. 22.	731 ft. W. 37° N.	.....	6.06	
B	March 28 to 31.	84 ft. W. 39° N.	1.53	10.54	Analysis "B, March 31 to April 22" was lost.
C	March 28 to 31.	675 ft. N. 44° W.	0.72	6.8	
C	March 31 to Apr. 22.	675 ft. N. 44° W.	.....	7.16	

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III. Analyses of Vegetation.

This investigation follows the lines of the most important researches made on the subject of smoke-injuries during the past 25 years in Europe and this country. The key to an interpretation of the results may be thus shortly stated:

1. After an immense amount of experimentation with plants growing in their native places, and with those transplanted to research laboratories, it is universally conceded by all competent investigators that the amount of damage to plant-growth from all other inorganic causes is a negligible quantity compared to that by the vapors of gases of HF, HCl, oxides of nitrogen, and especially oxides of sulphur.

2. Except in a few isolated cases, all of these but the last may be neglected.

3. Even in the vicinity of metallurgical works the amount of injurious gases evolved from the fuel is much greater than that produced by the metallurgical processes.

4. The ultimate source of the injury is to be sought in the oxidation and volatilization of the sulphur existing largely as pyrite and other sulphides in the coal. Wood-fuel is entirely free from this noxious ingredient.

5. The injurious action of the sulphur oxides which ultimately become acids, and which are thrown into the atmosphere by the combustion of pyritiferous coal, is intensified by the imperfect combustion of the coal, and the consequent production of soot, a greasy hydrocarbon which becomes saturated with the acid and keeps the spots on which it is deposited constantly moistened by this most deadly of poisons to vegetation.

6. A percentage of over 0.30 SO_3 in the ashes of the needles of evergreens, or leaves of deciduous trees, has been found, in the German forests, to indicate an unhealthy condition, except where there is an unusually large amount of sulphur compounds in the soil. It is not safe to assume this limit as applicable to other and especially to distant regions, but it may be safely held that 0.5 per cent. of sulphur trioxide in the needles, leaves, or twigs of plants is an indication that the plant is being poisoned by the absorption of the products of combustion in the air, through its organs above the ground.

7. Owing to the extreme solubility of all the ordinary sulphur oxides in water, and the free circulation of water in the

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ground, the poisoning of vegetation by sulphuric acid through its roots, in any case not artificially arranged for the purpose, may be dismissed as highly improbable.

8. Where the external parts of plants are constantly exposed to this poison through years, the percentage in the leaves shows a slightly accelerated increase with their age. Younger plants with more vitality, on the whole, show fewer symptoms of disease and approaching death than older plants, because they are better able to survive the successive impairments of the leaf-organs, and also to eliminate the active cause of destruction.

9. For the purpose of ascertaining how far a local source of contamination of the atmosphere is responsible for the injury to the vegetation in an adjoining district, it is necessary to determine: (*a*) that the injury is unquestionable; (*b*) that the percentage of SO_3 in the external organs of the plants is above normal; (*c*) that this is not due to the absorption of sulphates from the soil by the roots (some of which, like calcium sulphate, are nutritious and of direct benefit); (*d*) that vegetation of the same kind on practically the same soil, but more distant from the suspected source of contamination, contains a smaller percentage of SO_3.

In Table II. (page 511) these items are given.

Altogether fourteen specimens of vegetation were chosen for analysis, as follows: 1. Young hemlock hedge near green-house, 10 years old. (Position marked on the map.) 2. Hemlock hedge 14 ft. high, 50 years old, near green-house. (Position marked on the map.) 3. Norway spruce nearest to furnace and unprotected. (Position marked on the map.) 4. Hemlock from road running through woods west of house in direction of Cummins' Run, Airdlie, near Milford, Pa.; Apr. 15, 1905. 5. White-pine needles from young tree 50 ft. N. of house, Airdlie, Westfall township, Pike county, Pa.; Apr. 15, 1905. 6. Young white-pine from corner of woods at entrance of wood-road west of house, from Airdlie, near Milford, Pike county, Pa.; Apr. 15, 1905. 7. Young hemlock (± 4 years) from Sap Swamp Meadow, ± ¼ mile N. Edgemere Club House, Pike county, Pa.; Apr. 16, 1905. 8. Spruce trees from near Edgemere Club House, Pike county, Pa.; Apr. 16, 1905. 9. Spruce from Sap Swamp Meadow, ¼ mile NE. of Edgemere Club House; Apr. 16, 1905. 10. White-pine from Sap Swamp

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TABLE II.—*Analysis of Vegetation.*

No.	Date of Collection, 1905.		SO_3 in Needles or Leaves.	Ash in Needles or Leaves.	SO_3 in Ash.	SO_3 in Soil.
1.	March 31.	*a*	0.813*d*	3.42	23.77	.....
	Sept. 16.	*b*	1.05	4.39	23.92	.....
	Oct. 21.	*c*	0.88	.....	.....	.....
2.	March 31.	*a*	1.03	3.596	28.64	.....
	Sept. 16.	*b*	1.01	4.95	24.94	0.1
	Oct. 21.	*c*	0.86	.....	.....	.....
3.	March 31.	*a*	1.08	7.11	15.19	.....
	Sept. 16.	*b*	1.06	6.24	16.98	.....
	Oct. 21.	*c*	0.99	.....	.....	0.34
12.	Sept. 12.	*b*	2.35	9.78	24.01	.....
	Oct. 21.	*c*	1.61	.....	.....	.....
14.	Sept. 12.	*b*	1.66	8.23	20.17	.....
	Oct. 21.	*c*	1.98	.....	.....	.....
4.	April 15.	*a*	0.296	3.05	9.7	.....
	April 15.	*b*	0.47	3.89	12.08	0.08
7.	April 16.	*b*	0.42	3.35	12.53	0.07
13.	Sept. 12.	*a*	0.60	3.64*e*	16.48	0.05
	Sept. 12.	*b*	0.84	4.27	19.69	0.5

a Analysis by John Frazer.
b Analysis by J. K. H.
c Analysis by C. F. M.
d Mean of two determinations.
e Mean of three determinations.

No. 1, Needles of hemlock 10 years old; 2, Needles of hemlock 50 years old; 3, Needles of Norway spruce; 12, American beech, much exposed; 14, American beech, less exposed; 4, Needles of hemlock from Airdlie, Pike county, Pa., free from any contamination by the air; 7, Needles of hemlock 4 years old, from Pike county, Pa.; 13, Needles of hemlock from country-seat, ½ mile to the north of the industrial works under consideration.

Meadow, ± ¼ mile N. of Edgemere Club House; Apr. 16, 1905. 11. Pitch-pine from Sap Swamp Meadow, ± ¼ mile N. of Edgemere Club House; Apr. 16, 1905. 12. American beech from very near the NE. line of the industrial works before mentioned. (Position marked on the map.) 13. Hemlock from country-seat ⅘ mile N. of these works. 14. American beech on the lawn of the villa, but more remote than 12 from the works. (Position marked on the map.)

The numbers were given to the objects in the order of their analysis; but in the preparation of the table the analyses of the vegetation of the villa were put together. For this reason analyses Nos. 12 and 14 follow No. 3. Eight analyses were

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made of vegetation in Pike county, nearly a hundred miles distant: to wit, three from Westfall township, near Milford, and five from Delaware township on the high hills back from the river of that name. It was useless to record on the table all of these analyses of very remote plants, and therefore only two analyses (Nos. 4 and 7), one from each township, were selected as typical.

Nos. 1, 2, 3 and 12 are hemlocks and deciduous trees of the villa, exposed by the currents of air to the vapors and gases from the works, which are close to them. No. 14 is also from the villa, farther from the works and more sheltered than the rest by position. No. 13 is a hemlock similar to Nos. 1 and 3, and growing on similar soil, but nearly a mile distant, and therefore presumably less likely to suffer from the furnace-gases. Nos. 4 and 7 are plants selected from a distant part of Pennsylvania (Pike county), where contamination through the air is unsupposable.

From an examination of the data it will appear that all the analyzed plants from the villa showed much greater total quantities of SO_3 in the needles or leaves, and in the ash, than the hemlock No. 13, which was distant from the works. The quantity of SO_3 in the soil was also greater in Nos. 1, 2, 3, 12 and 14 than in No. 13, but it was so small in both cases that it could not have exercised any important influence on the results of the analyses.

The plants from Pike county (Nos. 4 and 7) are too remote to be of any value for comparison if there were any notable difference of sulphur compounds in the soil in the two localities; but as this is not the case, they offer a striking illustration of the difference of SO_3 content between the vegetation in an industrial city and that of a virgin forest, even when the natural supply of sulphates from the soil is nearly the same in both cases.

IV. Quantitative Determination of Sulphur in the Air.

The object of this examination was to fix the amounts of the plant-poison in the air during the prevalence of winds from various quarters, and to compare these amounts with others present when the wind was blowing directly from the works adjoining the villa.

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If it were found that the sulphur oxide percentage in the air invariably decreased with the divergence of the wind from the quarter occupied by these works, the conclusion would be justified that the works were supplying the harmful agent.

But this question is always the most difficult to determine, and especially in large industrial neighborhoods. Thousands of industrial and domestic chimneys are pouring out furnace-gases, and it might be plausibly argued that it is not just to single out one of the very many sources of these gases to bear the entire blame of pollution.

In this particular instance, other establishments lie around the southern skirt of the nearest works. In addition to this, several main and auxiliary lines of railway add to the contamination of the air. It would seem at first sight very difficult to fix the responsibility where so many offenders are concerned.

During the observations here tabulated, the locomotives on the various railroads above mentioned did not vitiate the results by adding any appreciable quantity of sulphur oxides to the air taken for examination.

The plan of the experiments was to get successive couples of observations during the continuance of a wind, the one to windward, and the other to leeward, of the works under investigation. But as it was found impracticable to occupy stations to the south or east of the works, stations on the north and west sides were occupied, and these stations sufficed, because the house and conservatories are situated in the northeast corner of the property and an east wind could only bring its noxious burden to the outlying parts of the property to the southwest.

DA, D, E and F were four stations just north of the extreme NE.–SW. boundary; stations B and I were immediately north of them; and stations G and H on the west side of the works.

With a southeasterly wind, therefore, stations D and DA would receive a portion of the products of combustion of all the fires in the city to the southeast, and, in addition, those evolved by the works. At stations G and H, on the contrary, with this same wind, all the former would be obtained but none of the latter.

The collection of these sulphur oxide gases was effected by pumping a measured quantity of the air into a flask containing

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TABLE III.—*Determinations of SO_3 in Measured Portions of Air Examined.*

	Number.	Station	Date, 1905.	Grams SO_3	Grams p. cu. M	Remarks.
	VII.	B	May 29.	Analysis	rejec'd	Wind W. Very little smoke, but odor perceptible. 10,000 cc. air.
l	VIII.	B	June 9.	0.0017	0.17	Wind variable, generally SW. to W. No visible smoke over lawn. Half saturated solution. Fine aperture delivery tube. 10,000 cc. air passed.
l	IX.	B	June 9.	0.0003	0.03	Wind as in preceding. ⅔ saturation. 10,000 cc. passed.
m	X.	DA	June 15.	0.0004	0.04	Wind variable, very light. Shifted from SE. to SW. before commencement. 10,000 cc. air passed.
m	XI.	E	June 15.	0.0003	0.03	Wind shifting. Smoke poured over station after experiment was made. 10,000 cc. air passed.
n	XII.	D	June 17.	0.0029	0.29	Wind SW., variable and puffy. Cloudy sky.
n	XIII.	F	June 17.	0.001	0.10	Wind SW., variable and puffy. Cloudy sky.
o	XIV.	G	June 20.	0.0005	0.05	Wind easterly, variable. Smoke rose high.
o	XV.	F	June 20.	Analysis	lost.	Wind easterly, variable. Smoke rose high.
p	XVI.	D	June 26.	0.0002	0.013	Wind SW., steady. Odor of smoke perceptible. 20,000 cc. air passed. Mean of two determinations.
p	XVII.	D	June 26.	0.0	0.0	Wind W–SW. No smoke visible.
q	XVIII.	B	June 27.	0.0	0.0	Wind N. shifting to NW. Clear. Fairly steady.
q	XIX.	D	June 27.	0.0	0.0	Wind N. Clear. No odor.
r	XX.	D	June 30.	0.0032	0.16	Wind S. to SW., rather steady. Clear. Smoke visible on place. 20,000 cc. air passed through solution.
r	XXI.	G	June 30.	0.0	0.0	Wind S. to SW., rather steady. Shifted slightly to S. Passed 10,000 cc.
	XXII.	D	July 1.	Analysis	rejec'd	Wind S., steady. Cloudy. Considerable smoke and odor. 20,000 cc. air passed.
s	XXIII.	D	July 1.	0.0001	0.01	Wind S. to SW., cloudy. Odor of smoke noticeable. 10,000 cc. air passed.
s	XXIV.	B	July 1.	0.0	0.0	Wind W. of S. Cloudy. Wind light. 10,000 cc. air passed.
	XXV.	D	Sept. 12.	0.0	0.0	Wind SW. to W., variable. No smoke visible blowing on lawn.
t	XXVI.	D	Sept. 16.	0.0005	0.05	Wind SW. to W. At times smoke visible over station. Odor noticed.
t	XXVII.	H	Sept. 16.	0.0005	0.05	Wind SW. to W. At times smoke visible over station. Odor noticed.
	XXVIII.	H	Oct. 27.	Analysis	rejec'd	Wind NE. to E., variable. New pump tried for the first time: not in order. Result canceled.
u	XXIX.	H	Oct. 27.	0.0019	0.19	Wind NE. to E., variable. Old pump.
u	XXX.	I	Oct. 27.	0.0016	0.16	Wind NE. to E., variable. Old pump.
	XXXI.	J	Nov. 32.	0.0005	0.05	Wind S. 40° W.

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sodium carbonate solution. Generally 10,000, but in some cases 20,000 and up to 45,000 cc. of air were taken.

The following records are given in grams of sulphur trioxide contained in 100 cu. m. of air, this volume having proved the most convenient to avoid fractions without unduly increasing the record numbers. No amount smaller than 1 g. of SO_3 per 100 cu. m. was noted in any of the experiments.[1]

The winds were usually variable, and during the half hour or more necessary for a single experiment they frequently shifted, so that two determinations at the same point and immediately succeeding each other were not entirely alike. This variability is sometimes indicated by converging arrows platted from the two directions between which the variability was noticed; in other cases, where less accuracy was required, a waved line for the shaft of the arrow expresses it. The stated directions of the wind give the earlier direction first and the point to which the wind shifted last. It frequently happened that the direction of the air during the experiment passed from a quarter partly affected by the furnace-products of the works to a quarter entirely unaffected by them, as in the first experiment, shortly to be explained.

Analysis VII. was rejected.

For the purpose of facilitating a comparison of the results, and to avoid using letters which had been used for stations on the map, a letter was employed for both of the paired determinations, beginning with the letter "*l*."

Thus "*l*" was used for determinations VIII. and IX., made June 9 at station B, with the wind shifting from SW. to W. During the first experiment, while its direction more nearly coincided with that of a line to the works, the air proved to contain 17 g., while in the second case, where it had changed so as to clear the works entirely, only sulphur oxides sufficient to produce in the analysis 3 g. per 100 cu. m. of SO_3 were found. This pair of observations is marked "*l*" on the map.

The pair of observations "*m*" were taken, respectively, X. at B and XI. at E, on June 15. The wind shifted from SE. to

[1] Much if not the greater part of the sulphur oxides was carried by the air as SO_3, but was combined in its sodium salt as SO_3. In the statements of grams per 100 cu. m. is meant that the sulphur oxides of all kinds left by the air in its passage through the fixing-flask when converted to sodium sulphate represented so many g. SO_3 per 100 cu. m. of that air.

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SW. The selected unit volume showed 1 g. more sulphuric oxide when the direction was most nearly from the works.

A much clearer proof of the responsibility of the works is found in the paired observations "*n*" (XIII. at D and XII. at F). With a variable SW. wind, part of the combustion-products of the western works would be carried to D, but not to F with the same wind. The difference is very marked; there being at F one-third of the amount of SO_3 which was detected at D.

Experiment XVI. was made on June 20, and was isolated on account of an accident which happened to the analysis of its companion, No. XV. It shows that with an easterly wind passing very high over the works, 5 g. of SO_3 per unit volume was present.

Couple "*p*" consists of observations XVI. and XVII., taken on June 26, at station D; the wind in the first being SW. and carrying 1 g. SO_3 from the contaminating quarter, while the second, with a more westerly wind, showed no sulphur-compound.

Couple "*q*" (XVIII. at B and XIX. at D) gives important information, clearing the part of the city north of the villa from inculpation in the injury to vegetation. With a north wind both records are 0.

Couple "*r*" (XX. at D and XXI. at "G") is very instructive. With a SW. wind the air showed 16 g. SO_3 per unit volume, whereas with the same wind at G, west of the works, the content of SO_3 was 0.

Analysis XXII. was rejected.

Couple "*s*" (XXIII. at D and XXIV. at B) is equally significant. With a SSW. wind, at D the air showed 1 g. SO_3 per unit volume, whereas at B with the same wind it showed no sulphur-content.

XXV. is an isolated analysis at D. Wind not from works. No sulphur oxides.

Couple "*t*" (XXVI. at D and XXVII. at H) showed each 5 g. per unit volume, with the wind varying from SW. to W.

Analysis XXVIII. was rejected.

Couple "*u*" (XXIX. at H and XXX. at I), with the wind NE. to E., variable, showed 19 g. and 16 g., respectively. As would have been expected, there was a larger percentage of sulphur oxides in the air at the former than at the latter sta-

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tion, because these winds would carry more of the furnace-gases to H than to I.

From all these observations it follows that the acid gases in the atmosphere which have killed and are still killing the plants at the villa, come in largely preponderating amount from the adjoining works.

The injury to the vegetation of the villa by possible furnace-gases emanating from any direction of the compass except from the quadrant between east and south (in which the works lie) may be practically neglected.

The lines indicating the direction of the wind observed at each station, with the number of the observation, and the percentage of SO_3 determined, will be seen upon the chart. The accompanying letters and figures will be easily understood.

Thus, "B. 6.9 VIII. 17" means that at station B (where the note will be found, the direction of the wind being indicated by the arrow), on June 9, experiment VIII., the air contained 17 g. of SO_3 per 100 cu. meters.

Where fewer figures and letters are used, the Roman numeral stands for the number of the experiment, the letter for the station, and Arabic numerals for the number of grams of SO_3 which the air contained per 100 cu. meters.

V. Examination of Soot Emitted by the Works.

The proof of the existence of soot in the atmosphere and of the principal source from which it is supplied is extremely easy. On any day when the wind is from the direction of the nearest works large volumes of unconsumed hydrocarbons may be seen pouring out of certain stacks, may be traced through the air in their course, and occasionally may be seen and felt in the act of depositing themselves on the person of the observer.

Cloths were suspended in different parts of the property and were soon covered by the mixture of unconsumed carbon and hydrocarbons from the combustion chimneys, with metallic oxides (usually ferric oxides) from the process stacks. If the cloth were white it changed in a day or so to gray, and in spots to black. Every plant and permanent fixture of the villa exposed to the wind from the works is covered with this deposit.

On Sept. 26, pieces of white paper were pressed against the stalks of the grass in the fields and drawn rapidly upwards.

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There was left a deep black mark, as was the case when one of the stone balusters or the rail of the steps was wiped with a piece of white paper.

Even the objects inside the house were not protected by the closed doors and windows. The doors and windows of the drawing-room, during the absence of the occupants in July and August, were constantly closed; yet white curtains which had remained suspended there during this time were blackened and soiled.

That the soot came almost exclusively from the nearest works and was largely due to improper firing is proved by the fact that on one occasion when the smoke was unusually annoying, during a visit of one of the then officers of the works to the villa, he gave such directions that on the following day and for some time thereafter the nuisance was very much abated.

In order to ascertain as nearly as possible how much soot was being carried over the place and how it was being deposited, Mr. John Frazer prepared eight porcelain crucible-covers, placed on the inner surface of each a layer of vaseline, and afterwards weighed them with the additional substance.

These were numbered consecutively, and placed as follows: No. 1, on the west corner of the main porch under the roof; No. 2, on the east side of the main porch; Nos. 3 and 4, on the leeward side of the house, No. 4 nearest the kitchen; No. 5, on the south side of the portico roof of the small house (see map); No. 6, on the north side of the small house under the eaves; No. 7, on the south side of the cart-shed; No. 8, on the north side of the cart-shed.

The area of each of these porcelain crucible-covers was very closely 14 sq. cm. (2.17 sq. in.).

They were collected on Oct. 7, eleven days after they had been put in place.

The covers, first weighed, to ascertain by difference the amount of added soot and dust, were then incinerated and weighed a third time to determine, by subtraction of the weight of the original crucible-covers, the amount of the incombustible matter or dust; this weight added to the weight of the vaseline and subtracted from the second weighing, gave the amount of the soot.

The results are shown in Table IV.

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TABLE IV.—*Amounts of Soot and Dust Deposited in Eleven Days.*

	Combustible Matter. (Soot.)		Incombustible Matter. (Dust and Ash, principally Fe_2O_3.)	
	Deposited on 14 Sq. cm.	On 1 Sq. meter.	On 14 Sq. cm.	On 1 Sq. meter.
	Grams.	Grams.	Grams.	Grams.
No. 1.....	0.0012	0.857	0.0010	0.714
No. 2.....	0.0008	0.571	0.0012	0.857
No. 3..... ...	Lost.			
No. 4.....	Lost.			
No. 5.....				
No. 6.....	0.0004	.....	0.0005	
No. 7.....				
No. 8				

This means that in the course of eleven days from 0.5 g. to 0.8 g. of soot were deposited on every square meter over the villa.

It is not safe to ascribe all the incombustible matter to the establishment examined, but certainly the larger part of it, which consists of ferric oxide, may be thus ascribed; for this compound is one not frequently met in the impurities of ordinary air, but is quite common in the neighborhood of works engaged in the manufacture of iron and steel.

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Bibliography of Injuries to Vegetation by Furnace-Gases.

Summaries and Synopses of Treatises Arranged in Chronological Order, to Accompany my Paper, "A Search for the Causes of Injury to Vegetation," etc., read by title at the London Meeting, July, 1906.

BY PERSIFOR FRAZER, PHILADELPHIA, PA.

(New York Meeting, April, 1907.)

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1. SMOKE PREVENTION. Report of Select Committee of House of Commons (1843).

Nuisance considerably abated in Leeds (Wm. Backerd, July 13, 1843, 239 pages).

A synoptic index, p. 211, gives the digest of every subject taken up.

2. FUEL: ITS COMBUSTION AND ECONOMY. D. Kinnear Clark, C.E. 24mo, 354 pp. Crosby Lockwood & Co., London (1880).

Includes abridgement of treatise on combustion of coal and prevention of smoke, by C. W. Williams, A.I.C.E.

3. DIE RAUCHSCHÄDEN IN DEN WÄLDERN DER UMGEBUNG DER FISCALISCHEN HÜTTENWERKE BEI FREIBERG. Dr. J. v. Schröder, Professor in Tharand, und Dr. A. Schertel, Vorstand des Hüttenlaboratoriums zu Freiberg. Royal 8vo, 27 pages, 19 pages of tables and a map. Ernst Maukisch, Freiberg (1884). (Aus dem *Jahrbuch für das Berg- und Hüttenwesen*, 1884.)

In 1861, Reich and Stöckhardt examined the extent of injury to vegetation by the Muldner works.

Needles and twigs of pine showed in 100,000 parts: Pb, 5 to 50; As, 3.3 to 14.3; SO_2, 62 to 120. In the top twigs, 5 to 8; needles, 10 to 16; thicker twigs, 10 to 22; thinner twig-ends, 17 to 54 of lead. Lead greater in amount in sickly twig-ends with few needles; also in the bark than in the naked wood of the trunk, which had hardly a trace; also in the part turned toward the works than on the opposite side. Pb and As diminished in the vegetation with distance from the smelter; and the same with regard to the soil. Fresh snow was collected, and SO_2, As, and Pb detected in it.

The authors showed that after repeated smoking with soot and arsenic-vapors, and sprinkling with white-lead, the growth of the plants was not injured. Their conclusion was that to the SO_2 in the smoke and the metallic poisons in the soil must be ascribed the injury to vegetation. The chronic poisoning by SO_2 is through Stöckhardt's experiment on pine-trees indubitably proven. Nobbe experimented on vegetation in earth treated with solutions of As and Pb and mixed with these salts. These investigations, as well as the results of other experiments, are given in the work of Schröder and Reuss.

Reuss has shown that the general injury as well as its intensity can be measured by the entire content of H_2SO_4 in the organ of the leaf; or rather the surplus of H_2SO_4 in the injured over that in the uninjured organs of the same plants in the same region.

The normal percentage of healthy pines from regions adjoining the smoke-affected area was 0.162 per cent. That in the area in which injury is just observable to the eye was 0.210 to 0.300 per cent.; in the second (higher) grade of injury, 0.3 per cent. to 0.5 per cent.; in the third, 0.5 and upwards.

The increase of H_2SO_4 is not due to the sulphates deposited on the needles by the floating dust. The content of H_2SO_4 in the ash of healthy regions was 5.47 per

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cent.; in those of the first grade of injury, 7.67 per cent.; in the second, 10.65 per cent.; in the third, 17.09 per cent.

The H_2SO_4 determination was made in the following way: The needles freed from all the twigs, having been dried and well powdered, were stirred to a thin decoction in a solution of sodium carbonate and distilled water in a platinum dish, and evaporated to dryness. The residue was charred (*verkohlt*) and treated with water. The leached coke was then completely incinerated. The ash was then united with the extract, the solution evaporated, and treated with an excess of HCl and precipitated (after precipitation of the silica) by $BaCl_2$.

For determination of the metal-content 100 g. of the dried needles were treated with dilute HCl, the solution treated with $KClO_3$ in moderate heat, to decompose the organic substances, filtered, and precipitated by H_2S. The precipitate was then used to determine in the usual way As, Pb, and Cu. The undissolved residue from the extraction of the needles was carefully reduced to ash in the muffle-furnace, treated first with Na_2CO_3, and, after thorough washing, with dilute HNO_3, the solution precipitated by H_2S, whereby a further portion of Pb was precipitated, and the filtrate added to the filtrate of the first sulphhydrogen precipitate, to determine the zinc oxide.

The maximum percentages of H_2SO_4 found by Schröder and Schertel in the needles of pine-trees unquestionably healthy, not too near the smelters, and which certainly had not previously been subjected to smoke, were as follows: 0.204, 0.213, 0.223, 0.234, 0.226, 0.234 per cent. H_2SO_4.

No signs of injury are visible in vegetation showing less than 0.250 per cent. H_2SO_4. If one plat on a map the area within which a percentage of 0.230 H_2SO_4 is found, it is no longer a closed region, and one would see in the continuous forest-areas of Zellaer, Tharand, Struth and Freiwald small island-like patches of injured vegetation surrounded by healthy trees.

All the forest-areas in which the needles contain 0.250 per cent. and upwards of H_2SO_4 can be inclosed by the periphery of an elliptical area of which the main axis runs northwest and southeast. South of the Mulda works, and on the left bank of that stream, the greatest damage has been done—maximum, 0.292 per cent. (much less than that of the Harz, which is 1.33 per cent.). On the road from Weissenborg to Lichtenberg, opposite the wood-manufactory in Berthelsdorf, the pines are sickly, with black misshapen needles, and 0.238 per cent. H_2SO_4. It lies exposed to the high chimney of the Mulda, but the manufactory chimney is the immediate and preponderating cause of the trouble. A test of trees in Butze's woods, though apparently uninjured, showed 0.442 per cent.

Near Gesegnetes Bergmann's Glück all the trees seem to be affected, but the percentage of H_2SO_4 is only 0.285. A test opposite the Münzener Hammer and the wood-planing mill shows 0.488, which is explained by a union of the effects of the Münzner, the Halsbrücke, and the locomotive smoke. It was not possible to draw concentric lines with the smelters in the center showing the more and the less injured areas, as has been done for the Oberharz, because the Muldner works are not inclosed by woods, and the chimney-gases are not so acid.

The percentage of metal oxides is not parallel with that of H_2SO_4. As, Sb, and Pb do not seem to poison the vegetation, as they are found in the healthiest patches. These metal oxides are deposited as dust on the needles and are therefore much affected by the condition of the weather—dry or wet, etc.

The metal oxides are not exclusively due to the furnace-gases, but may be derived from the sulphides, galenite and zinc-blende in the gneiss. Zn, Cu, As, etc., find their way to the vegetation through the weathering of the rocks.

The conclusions reached by the authors are:

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1. The areas affected by the furnace-gases are certainly defined. Beyond these limits the injurious action will not extend in the future because constantly increasing provisions for condensing the noxious constituents are being made.

2. In the district affected by the Muldner works the injury not only has been diminishing since 1865, but also since the taxation by the Committee in 1876.

3. Münzner's machine manufactory and wood-planing works in Obergruna has its part in the injury by furnace-gases which appears in the Mulda valley below Hohentanne and to the confluence of the Mulda and Bobritzsch.

Though it is very difficult to express in figures the relative responsibility of the different sources of contamination, it is unfair to consider only the action of the smelter furnace-gases.

4. Smoke-Nuisance Abatement (Metropolis) Bill. Report of Select Committee of the House of Lords, with the Proceedings of the Committee and Minutes of Evidence. 321 pp. Eyre & Spottiswoode, London; Adam & Charles Black, Edinburgh, and Hodges, Figgis & Co., Dublin.

Mr. Wm. R. E. Coles, Engineer appointed by the Home Secretary to examine furnaces, in the metropolis, complained of for causing smoke nuisances: I report to the Commissioner of Police. . . . Also am Honorable Secretary to "Smoke Abatement Institute."

Prevention or mitigation could be attained from the nuisance due to private houses by alteration of the structure of the grates, and by change of the fuel.

By the first, the new coal is put under the fire instead of on top. Better still is the method of altering the draft so that the burning takes place at the top and the combustion-products are carried down to the flues which connect them with the chimney.

The second method is that of using anthracite coal instead of bituminous. There is a vast quantity of anthracite coal in Wales. This or coke or gas could be employed. Mr. Davis, an Inspector, found when he used coal for heating and cooking and gas for lighting he consumed 51,000 cu. ft. of gas and 22 tons of coal per year. Another year, using gas for cooking and lighting and gas-coke for heating the rooms, he required 53,500 cu. ft. of gas for lighting, and 34,500 for cooking. The first year the coal and gas bills amounted to £24 11s. 6d., and in the second £16 18s. 3d. The only alteration necessary in burning coke instead of coal was to have fire-brick backs and jambs instead of iron.

Mr. James Edward Davis: Legal adviser appeals by Home Secretary to the commissions. Evidence chiefly on application of Acts of Parliament to London.

Supdt. Cutbush: Evidence chiefly concerning administrative and legal aspects of the question.

Ernest Hart, Chairman of Council of National Smoke Abatement Institute: Description of "saucer-feed on under grate-bar," Sir Wm. Siemens's coke and gas together. Sir Wm. Siemens has said the ideal of smokelessness is gas fuel. Earl of Harrowly asks: "Were you in London at the time Lord Palmerston made his great movement against smoke?" . . . In the last 20 years smoke has made a perceptible difference in the health of the inhabitants. Roses could be grown at Princes' Gate, but that is now impossible. Roses could grow in Kensington Gardens; now the very last conifer is dying or dead there. . . . Seven of the best physicians agree as to the enormous increase of mortality owing to the smoky fogs. . . . A short time since the Thomson furnace was brought to the notice of the Council, and the engineer, Mr. D. K. Clark, C.E., was instructed to make a series of scientific tests of the apparatus. The tug "Alexandra" was fitted with the furnaces and ran from Temple pier to Richmond, lay there two hours and returned to Westminster without products of combustion being visible at the top of the chim-

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neys except for about a minute each time the furnace was stoked, when a little pale smoke was emitted. When the fires had got low they were stoked with coal-dust, which produced smoke for two minutes, but not of dark shade. The owners report a saving of 20 to 25 per cent. in fuel burnt. The sister tug to the "Alexandra" was passed several times burning identical coal in an ordinary furnace and emitting volumes of black smoke. . . .

It costs £28,000 per year to clean the surface of the House of Parliament, and Westminster Abbey is falling to pieces in the same way. . . .

Coke is cheaper than coal. . . . One can roast by gas.

In the time of fogs (black, artificial fogs), the rate of mortality rose to the rate in the great cholera year; the annual mortality to 40 in a thousand.

The London fog goes like a great wall down to the Surrey Hills.

5. Zinc and Lead Content of Burnt Blast-Furnace Gases. Albert Vita, *Zeitschrift für angewandte Chemie*, p. 69 (1890).

Since 1887, the gas-cleaning apparatus has been much improved. No amount of cleaning will suffice to prevent material losses of zinc and lead.

The furnace-gases were drawn by aspirator through five Bunsen wash-bottles filled in the order given with water, HCl, HNO_3, HBr, and alcohol, respectively, the last for the absorption of Br vapor. At each test not less than 82 liters were thus passed through. For one ton of iron produced an average of 13,296 cu. m. of burnt gases was developed; 3.89 kg. for 1 ton iron, etc.

6. Condensation of Carbon Particles in Smoke. Robert Irvine, F.C.S., *Journal of the Society of Chemical Industry*, vol. ix., p. 1110 (1890).

The author erected a glass structure provided with two iron plates, each with points. All parts of the surface but the points were covered with shellac varnish. On admitting a mass of smoke (from pitch-oil), so thick that a bright light placed at the opposite side of the chamber was completely obscured, he passed a current between the plates by a small dynamo. The effect was instantaneous, and the chamber was cleared of smoke almost entirely. If a smoked glass be examined under the microscope each particle will be seen to consist of amorphous carbon surrounded by an areola of oily matter. Rain does not precipitate it, on account of its water-proof covering, but air-currents which drive the particles together clear the fog off suddenly.

(Illustration of a smoked microscopic slide showing soot-particles with "areola" of hydrocarbons surrounding them.)

If commercial lamp-black or soot from imperfectly burned coal be treated, empyreumatic matter is driven off as a brown greasy substance consisting of crysene, pyrene, capnomor, etc., which cause lamp-black to cohere like a damp snow-ball when pressed.

In a newly made or mended fire only light blue and yellow-brown colored vapors are given off, consisting of solid or liquid hydrocarbons, which pass into the atmosphere. As the coal becomes heated, these burn with a smoky flame. At this stage, owing to the imperfect combustion of these gases, finely divided particles of carbon are formed and black smoke is added to the greasy volatile matter, making black or brown fog. Although the soot is only 3 per cent. of the mass, it powerfully obstructs the light. When the black fog disappears, without being blown bodily away, it is by the agglomeration of the minute soot-particles.

Rollo Russell estimates the coal consumed in London at 20,000 tons per diem, which at 3 per cent. makes 600 tons of smuts.

Mr. Elliott's washer draws the smoke from the stack by a fan and agitates it in a closed space.

There is no difficulty now in preventing smoke from any factory chimney.

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Macaulay, in a lecture at Liverpool, February, 1888, estimates the annual waste of coal in England at 45,000,000 tons, valued at £15,750,000 at the pit-mouth.

7. CONTAMINATION OF AIR BY COMMONER ILLUMINANTS. E. Cramer, *Journal für Gasbeleuchtung*, vol. xxxiv., p. 27 (1891).

Comparison of values of tallow and paraffine candles, gas, etc., for standards in photometric experiments and heat-units, etc.

8. THE ANALYSIS OF THE AIR OF LARGE CITIES. Editorial note on the Committee (Drs. Bailey, Cohen, and Tatham and Mr. P. J. Hartog), appointed by the Town Gardening Section of the Manchester Field Naturalists' Society. *Industries*, vol. x., p. 91 (1891).

The Section planned simultaneous analyses from seven different stations in Manchester and Salford to ascertain :

1. The comparative purity of air in densely and in sparsely populated districts.
2. The relation between atmospheric impurities and prevalent sickness and death.
3. The amount and distribution of noxious ingredients specially injurious to plant life—*e. g.*, SO_2.
4. The extent to which smoke and noxious gases are due (*a*) to dwellings, (*b*) to factories.
5. The nature of fog, and chemical character of air during the prevalence of fogs.

Dr. Bailey reported experiments had commenced. Snow carried to ground large quantities of HCl and H_2SO_4, also some elements of sewage. Depositions on leaves were in amount proportionate to population. The greatest injury to plants was due to emanations from dwelling-houses. He estimated that two tons of "blacks" and H_2SO_4 were deposited per sq. mile of city area.

9. THE NOXIOUS VAPOURS OF MANURE-WORKS. John Morrison, *Journal of the Society of Chemical Industry*, vol. x., p. 338 (1891).

Before the Alkali Act of 1881, the conditions connected with the manufacture of superphosphates from manure were frightful. Now it is much better. The noxious vapors must be distinguished from the powerfully offensive and searching stinks from bone-boiling and blood, fish, and flesh-offal treatment. Stifling fumes of mineral phosphates treated with H_2SO_4 consist of fluoric, carbonic, and sulphuric acids, and steam, and unpleasant oily vapors. On the Tyne four or five tons of HCl acid would be discharged from South Carolina River phosphate in a week. I have devised most of the apparatus for arresting manure-works vapors. These vapors contain all the ingredients necessary to deposit themselves, and but one condition—heat—to retard this. For this reason, opposed to use of steam, I cool by flues with baffling diaphragms. This takes out the solids. The gases traverse one or more water-towers, wet-scrubbers packed with wedge-shaped wood-spars, before emerging at the chimney. The draft is a forced draft by exhaust-fan.

In the discussion, Mr. Morrison said he preferred a flue of about 200 ft. between the mixers and scrubbers.

10. INSPECTION OF CHEMICAL-WORKS. Mr. Brunner, M.P., *Journal of the Society of Chemical Industry*, vol. x., p. 637 (1891).

In the discussion in the House of Commons on the local Government Board vote, Mr. Brunner said the law was in an anomalous condition. A number of alkali-works are prevented from discharging noxious vapors, but many smaller works using the products made at the larger are not prevented. If the Inspector can induce the smaller works to consume these noxious gases then they come under the Act.

11. THE MANUFACTURE OF IRON IN ITS RELATION WITH AGRICULTURE. Sir I. L. Bell, *Journal of the Iron and Steel Institute*, vol. xlii., No. 2, p. 11 (1892).

The nutrition of plants is chiefly from the atmosphere. More carbon is stored there than in the crust in all forms. The ash of plants is 2.5 per cent. of weight,

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and of this 2.5 per cent. is iron oxide. Annually a human body extracts only enough iron to make a wedding ring. One three-hundredth by volume H_2S introduced into the blood would interfere with the iron changes and the whole would be vitiated in 25 seconds. If all the H_3N gas in the atmosphere were collected at sea level and at the atmospheric pressure it would not be thicker than ½ in. When fossil coal is thrown upon the fire all the hydrogen it contains unites with oxygen in forming water. Each ton of coal burnt gives 90,000 cu. ft. gas and 4.38 lb. NH_3 gas.

The paper is chiefly concerned with geological and chemical questions.

12. An Improved Method and Apparatus for the Removal of Smoke and Fog, Etc. Patented by E. Oades, Wokingham, Eng. *Journal of the Society of Chemical Industry*, vol. xi., p. 233 (1892).

Two sets of pipes or mains are laid under a roadway provided with suitable traps or gratings, through which air or fog is drawn and conducted to adjoining hearths or fires, etc.

12a. Improved Apparatus for the Purification of Gaseous Fumes, Etc. Patented by M. F. Purcell, Dublin, and G. Purcel, Los Angeles, Cal. *Journal of the Society of Chemical Industry*, vol. xi., p. 1025 (1892).

Patent apparatus for purification of gaseous fumes, air, etc. An ordinary exhaust-fan, into the casing of which numerous nozzles are fitted. Water, containing or not suitable substances in solution, is made to impinge on the vanes of the revolving fan, and is converted into mist and takes out all solid particles.

13. Smoke-Consuming Apparatus. A. R. Sennett, *Report of the British Association for the Advancement of Science*, p. 880 (1892).

14. A System of Purifying Smoke from Domestic and Other Fires. Col. E. Dulier, *idem*.

Mixing smoke as it leaves the boiler with a small quantity of steam generated in kitchen-range boilers. The mixed gases rise into an open chamber, the top of which is provided with pipes, placed in the direction of the prevailing wind, through which air passes and cools the gases. At the extreme top, just before entering the air, the gases are met with a spray of water from perforations in a conducting-pipe. The result of this treatment is a washing of the smoke and complete removal of all soot, dust and SO_2.

The amount of steam is small. At the Sloane gardens the expense is ten gallons of water per hour. The apparatus treats the smoke from a large kitchen-range burning 20 lb. coal per hour. The draft is not sensibly impaired (?).

15. The Identity of Lung-Pigment with Soot. J. Wiesner, *Monatshefi*, vol. xiii., p. 371 (1892).

Ueber den mikroskopischen Nachweis der Kohle in ihren verschiedenen Formen und über die Uebereinstimmung des Lungenpigments mit der Russkohle.

In the course of investigations under the microscope Dr. Wiesner has often been puzzled to determine whether small black particles, incapable of other examination, were coal or not. Various bodies are called coal, such as soot, charcoal, lignite, anthracite, and graphite.

In the interest of . . . the microscopic examination of atmospheric dust, in differentiation of writing-characters, etc.

On this occasion the author communicated the results concerning the problem stated, and concerning the pigment in the human lungs. Examination was made of the inks of the characters on papyri, mummy-bands, and the oldest known papers. No difficulty was found with ferro-tanno-gallates, but much with soot and India ink. A mixture of chromic acid and sulphuric acid was made in such proportions that in the reduction of the chromic acid the resulting chrome oxides were held in solution. For this purpose he treated cold concentrated potassium bichromate with excess of

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sulphuric acid, and added only so much water as was necessary to keep the separating chromic acid in solution.

The amount of sulphuric acid added should be just enough to keep in solution the whole of the chromium oxides separated from the chromic acid, but an excess of H_2SO_4, according to Dr. Wiesner's experience, is not injurious, nor is the potassium bisulphate. All cells are destroyed by this reagent, and carbon-particles are unchanged for a long time.

For years the author had noticed that soot is but little attacked by chromo-sulphuric acid. The little carbon-particles in the soot were in weeks or months hardly affected while the tarry mass went into solution. The particles of soot after a long time were unaltered, suspended in the liquid, and this fact led to a method for distinguishing soot in writing-inks. Part of a letter is cut out of the MS., placed upon platinum foil, and treated with chromo-sulphuric acid. The paper will entirely disappear, leaving only the writing-substance. If the fluid be removed and this residue heated on the foil it burns, leaving behind an ash, and proving carbon.

By this reagent lignite can be distinguished from charcoal, soot, and graphite.

Dr. Wiesner sets forth the results of a microscopical examination of the principal varieties of coal:

Amorphous Carbon.—The first experiments were with charcoal. No change was observed after it had been long exposed to the action of chromo-sulphuric acid, but the solution which came away from it was greenish.

Soot was not changed for a long time, but a slightly greenish tinge was imparted to the reagent, and the soot-particles disappeared after weeks.

Amorphous carbon is therefore in a state of extremely fine division.

Not infrequently one hears of transparent carbon.

This arises from an erroneous interpretation of F. Schulza's observation (*Nat. Vers.*, Gotha, 1851). The finest breath of soot on a glass plate looks translucent, but under the microscope it consists of isolated opaque points. Either of two explanations is possible. Either the very thin coal particles are brown or the reagent has by oxidation produced another compound which is brown.

Soot.—A layer of soot, not too thin, on a glass plate, under a high microscopic power shows two substances: one small black opaque points, and the other fine colorless, yellowish, or brownish objects lying together but more or less coalescent. If treated with Dammarlac, or oil of cedar, all but the black opaque particles disappear.

The atmospheric soot presents another aspect. Of course the fluid constituents are wanting, but several different kinds of solid bodies are present: small, apparently homogeneous, soot-particles; dendritic aggregates of them; black and brown fragments, generally of irregular, seldom of rounded form. The black fragments consist of black soot-kernels more or less united, yet under the highest magnifying power the cementing substance is not visible. The brown fragments are more compact and consist of a brown matrix in which small black soot-grains are imbedded. The matrix is derived from the liquid components.

If this soot be subjected to treatment in cedar oil, the large fragments disintegrate gradually into small roundish black balls, and dissolve finally into countless extremely small grains, which are identical with the above-named black bodies. In chromic acid, under the microscope, the soot remains some time before the same disintegration is effected. After weeks of subjection to this acid the particles are still visible, but smaller. But it is not possible to say from microscopic observation whether they have been partially dissolved or only still further disintegrated. At last after months they disappear entirely. [Then it is probable they were dissolving.—Ed.] Atmospheric soot contains, besides the coal-particles and cement, also

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masses of small bodies derived from the material used to produce it. Iron was thus found.

Brown Coal (*Lignite*).—However variously these substances appear to the eye, in the microscope they all agree in that the powder is quickly changed by chromic acid into a yellowish and finally colorless mass, which for the most part represents a skeleton or nexus consisting, as the reactions teach us, of cellulose. For if this detritus be washed carefully with water it is soluble in cupric oxide-ammonia, and becomes violet with chlor-zinc-iodine solution.

Gas is evolved from powdered brown coal by chromic acid and in a few hours the solution becomes green.

Anthracite.—If finely powdered anthracite be covered by Cr_2O_5 in a test-tube, after some days or weeks the color of the liquid will be changed into brown or green, according to its quality.

Welsh anthracite took eight days to show this change of color and behaved like pure carbon.

Under the microscope without exception the greater part of the substance consisted of black opaque particles; but there were also some brown grains, the easier to find the greater the proportion of oxidizable substances (not mineral impurities) it contained.

Such brown bodies were not found in brown coal, but nevertheless they may be there, as they represent a more advanced state of carbonization, in which either the entire cellulose of the plants from which the anthracite was produced is transformed into carbon compounds, or so little is present that the reagent has destroyed it.

Bituminous Coal (*Stein Kohle*).—Careful microscopic research proves that the variety last mentioned and this one are connected together by transitions.

Pulverized bituminous coal is rather quickly oxidized by Cr_2O_5; less so than lignite, but more so than anthracite.

Lignite dissolves without residue; the greater part of anthracite is unaffected; bituminous coal leaves a small residue, which behaves like the residue of anthracite, both chemically and under the microscope. The brown or reddish brown parts are divided into:

1. Bitumens (Harze)—*i. e.*, fusible bodies soluble in bitumen-solvents.
2. Bodies which react exactly like lignite to chromic acid.
3. Bodies which correspond exactly with the brown or reddish brown transparent anthracite grains in appearance and also in their behavior to Cr_2O_5. 1 and 3 are homogeneous, while 2 is often not so. The microscopic examination shows that bituminous coal is an intimate mixture of lignite and anthracite. Anthracite appears to predominate.

Charcoal.—As is well known, the percentage of carbon in a charcoal depends upon the temperature at which it is produced. At a comparatively low temperature there is produced the so-called red charcoal, which is advantageously used in the manufacture of gunpowder. At a high temperature black charcoal is produced, which is richer in carbon.

Powdered red charcoal reacts to Cr_2O_5 like lignite; black charcoal like anthracite.

Both red and black charcoal possess exactly the texture of wood. The outer cell-membranes of the former appear under the microscope brown, in the latter black.

Graphite.—Graphite powder covered by chromic acid was examined under the microscope for days, and indeed for two months, without the detection of the least appearance of diminution or solution even in the smallest particles. But there is an oxidizable constituent present, as the change of color of the supernatant fluid proves. It is hard to distinguish from soot, though it is not easy to reduce graphite

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to so fine a state of division. In combustibility, soot and charcoal-dust burn instantly on platinum foil; lignite, bituminous, and anthracite burn more slowly, but graphite is almost incombustible.

The following is the author's conclusion as to the identity of black lung-pigment and soot:

Virchow's "lung black" has been examined. Koschlakoff and Virchow have written on the subject. Traube proved that splinters of charcoal could enter the alveolæ. Knauff believed the pigment was composed of soot, from his experiments on dogs allowed to live in a smoky atmosphere. On killing them and examining their lungs the black lung-pigment was found to be indistinguishable from soot, but it was objected that this material might be the carbonization of extravasated blood.

Hoppe-Sayler says the pigments of the eyes, skin, melanitic carcinoma, hair, feathers, fish-bone, etc., are easily decomposed by treatment with alkali-solution or chlorine. But in the lungs and bronchial ramifications is a body of perfect black color which is insoluble in potash lye, and chlorine, and is therefore carbon.

From examination under the microscope of strips of lung-material with melanine spots, furnished by Prof. Sig. Exner, and from the behavior of this material when treated with Cr_2O_5, it was demonstrated that the pigment-particles of the lungs and of the sputum were of the same origin, and that they reacted to Cr_2O_5 just as the brown particles of atmospheric soot.

The black lung-pigment was compared with the black material in the discharge from the nose of inhabitants of smoky cities, and found to correspond.

Résumé.—1. The essential constituent of lignite is brown, and transparent; is made colorless by Cr_2O_5; and leaves, after treatment, a histologically determinable net-work of cellulose.

2. All other forms of carbon (anthracite, bituminous coal, charcoal, soot, and graphite) contain usually small quantities of a substance easily oxidized by Cr_2O_5, which turns it brown, and at last green. The residue is unalterable by long treatment with this reagent.

3. Anthracite is practically unalterable by Cr_2O_5, but contains a dark brown transparent body which is slowly oxidized by Cr_2O_5, leaving no cellulose.

4. Bituminous coal behaves under the microscope like a mixture of lignite and anthracite, and leaves a small residue of cellulose when treated with Cr_2O_5.

5. So-called red charcoal is thoroughly decomposed by Cr_2O_5, and, in a certain stage of the decomposition, cellulose in the form of wood-structure remains.

6. Soot freshly deposited on glass consists of small black carbon-particles which after weeks of treatment by Cr_2O_5 remain, and of fluid drops of oily nature. Atmospheric soot consists in part of fine carbon-particles, partly of aggregates of small particles either dendritic or irregular in form; or, less frequently, round fragments which either show black grains in a brown matrix or appear simply as more or less loose aggregates of black grains.

7. Black lung-pigment, which during the lifetime collects in every human lung, and especially in the interlobular connective tissues of the lung, and of which the true nature has not hitherto been sufficiently explained, consists of soot in the form of large or small dark bodies, which appear as fine, point-like grains, and after weeks of subjection to chromic acid show no alteration.

Melanines distinguish themselves from this pigment by the ease with which they are decomposed after a few minutes of treatment by chromic acid.

Note.—After this work was completed Dr. Wiesner received from Prof. Liebur chemically pure carbon prepared from soot obtained from a gas-flame on a cool porcelain vessel, and purified by heating to redness in chlorine, nitrogen and hydrogen. This substance contained 99.3 per cent. carbon.

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This body reacted very nearly like the black residues of anthracite and other coals described above, which were called amorphous carbon; but, probably on account of its extreme fineness, it was notably easier to oxidize by cold Cr_2O_5 than these residues.

Like the bodies described above as pure carbon, it is much more rapidly oxidized by chromic acid even under the boiling-point of water. Warmed over the open flame, the oxidation takes place with visible disengagement of gas.

16. AUTOMATIC APPLIANCE FOR RECORDING THE PRESENCE AND DENSITY OF BLACK SMOKE IN FACTORY CHIMNEYS. William Thomson, *Journal of the Society of Chemical Industry*, vol. xi., p. 12 (1892).

Two brass tubes about 5 ft. long, one of 1⅛-in. and the other of ⅝-in. internal diameter, are placed one within the other. A slit is made through both tubes lengthwise to within 3 in. of end. Another cut is made at termination of slit at right angles to first. A semicircular plate is made to join both tubes at the end of the 3-in. cut. Paper is made to move across the slit at the rate of 4 in. per hour and a copper tube inside the brass tube conveys cold water to keep the paper cool, it being found that smoke will not deposit so easily on a hot as on a cool surface.

The apparatus is then put into the chimney or flue, and the clock-work so attached as to keep the paper moving at a uniform rate of speed.

17. MODERN LEGISLATION IN RESTRAINT OF THE EMISSION OF NOXIOUS GASES FROM MANUFACTURING OPERATIONS. A. E. Fletcher, H. M. Chief Inspector under the Alkali Act, *Journal of the Society of Chemical Industry*, vol. xi., p. 120 (1892).

The Alkali-Regulation Act of 1863 was a new departure. It originally only dealt with hydrochloric acid. Its first effect was to heighten chimneys. This failing, the Gossage condensation-towers were tried. Operation nearly perfect. Standard condensation of 95 per cent. accepted by manufacturers. Not only acid character of furnace-gases was determined, but, by means of anemometer invented for the purpose, the volume escaping. Collapsable aspirator of vulcanized rubber used to take samples and absorbent of gas introduced into it.

Two-tenths grain of HCl in a cu. ft. of air, and not more than 5 per cent. of the total gas produced allowed to escape, together formed the restrictive law.

The consumption of a million tons annually in St. Helen's and Widnes made the consequent SO_3 from this and from the copper and glass works overshadow the evil which the Alkali Act sought to remedy.

In 1881 an amended Act superseding previous Acts was passed to adopt "the best practicable means for preventing the discharge into the atmosphere of all noxious or offensive gases evolved in such works." (Discussion on pp. 120 and 309 as to method of working of the two Acts.)

18. SMOKE-PREVENTION. Report of a Special Committee of the Engineers' Club, St. Louis. *Journal of the Association of Engineering Societies*, vol. xi., p. 291 (1892).

Detailed statement of the process of burning bituminous coal, with the changes which occur.

The great offenders as smoke-producers in large cities are boiler plants. Of 78 consumers in St. Louis, but 7 were using smoke-preventing apparatus. Nineteen have used some kind of smoke-preventers but have discarded them for various reasons. One found his smoke-consumer consuming one-eighth more coal than the common furnace.

The following is a condensation of Parts VI, VII and IX of the report:

VI. Requirements for a successful smoke-consumer.

1. Efficiency: (*a*) Development of high temperature; (*b*) regularity of action; (*c*) not easily got out of order; (*d*) small increase to operation.

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2. Capacity. Must be efficient when boiler is working to full capacity.

3. General applicability : (*a*) Ready adjustment ; (*b*) application in limited space; (*c*) low cost ; (*d*) few repairs ; (*e*) no injury to boilers, etc.

VII. Classification of the important types of smoke-preventing devices already proposed and the principles on which they depend.

A. Steam-jets to introduce air into the fire-place.

B. Fire-brick arches or checker-work.

C. Hollow walls for pre-heating the air.

D. Coking arches or chambers.

E. Double combustion.

F. Downward-draft furnaces.

G. Automatic stokers.

IX. Conclusions and Recommendations.

This deals with (1) a determination of the practical limits within which smoke emission may be confined, and (2) a determination of the applicability of various devices to the purpose intended.

Recommendations are as to legislation and the diffusion among the public of information as to the facts which may aid them in using smokeless fuel.

Various circulars and ordinances in Cincinnati and Pittsburgh.

19. THE COMBUSTION OF COAL IN HOUSE-FIRES. J. B. Cohen, Ph.D., and G. Hefford, A.I.C., *Journal of the Society of Chemical Industry*, vol. xii., p. 121 (1893).

The amount of sulphur in coal is of importance.

SO_2 in air oxidizes to H_2SO_4 and attacks stone, brick, and respiratory organs.

First smoke-abatement meeting, Leeds, 1842. House-fire principal cause of trouble. Amount consumed so small that it does not pay householder to alter systems of burning, which are very bad.

Corporation sewage-works at Holt (town near Manchester) use as fuel almost exclusively cinders from household fires.

1855, Delezenne estimated unconsumed carbon as 5 per cent. of the total weight.

Very black smoke, 0.1 per cent. of amount burned.

Analysis of Soot.

1. Manchester Air-Analysis Committee. Out-door deposit.
2. Roberts-Austen. Aspirated from flue.
3. Cohen-Hefford. Deposit in another flue.
4. Deposit in another flue.

	1	2	3	4
Carbon,	39	86–94	68.5	75.3
Hydrocarbons, etc.,	14.3	(H) 3.3–5.2	(H) 4.4	3.9
Sulphuric acid,	4.33	. . .	(S) 4.8	3.2
Mineral matter,	36.67	8–9.7	22.7	16.3

	House-fires.	Angus Smith. Black Smoke.	Cohen and Hefford. House-fire.
In flue-gases, grams per 100 liters,	0– 0.09	0.043	0.03
Per cent. C burnt (at 1.2 per cent. CO_2 in fuel-gases),	0–13	. . .	5.09
Deposited in chimney at 100 of coal,	0.61–2.25	. . .	. . .

Bailey showed that of sulphur in coal 53 to 55 per cent. escaped into air, 4.8 to 5.4 per cent. remained in the clinker, and 39.6 to 42.3 per cent. disappeared (!).

Angus Smith finds from coal containing 2 per cent. S, 0.23 g. SO_4 in 100 liters. From a direct determination of SO_2 in black smoke he found 0.07 SO_2 in 100 liters.

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Discoloration of Silver Articles.—This during foggy weather is attributed to formation of Ag_2S from S compounds in the air.

$AgSO_2$ is in air ; this was tried, but only produced blackening in presence of soot.

Air filtered through plug produced less effect than not filtered. H_2S is in very small quantity in air, but must be considered.

Mr. Thomson remarked that silver was blackened by HCl vapors. On repeating experiment it appeared that pure silver was not thus blackened, but alloys of Ag and Cu. The acid dissolves out the Cu as $CuCl_2$, which is later decomposed, leaving a black film of Cu on the silver.

In discussion the chairman (Mr. T. Fairley) remarked that leading the acid gases through alkaline solution was not sufficient to absorb them, but that they must be violently agitated in presence of the solution.

Thomson proved that HCl in the light produced more effect than H_2S.

If a glass plate moistened with glycerine were exposed much more soot was collected than from snow.

Dr. Lewkowitsch thought organic sulphur compounds were to be studied. He thought producer-gas was the best method to abate the smoke nuisance.

Prof. Smithells alluded to the doctrine that unburnt carbon and SO_2 furnished the air with valuable antiseptic media of infinite value to congested centers of population. He disbelieved it.

20. Town Smoke. Discussion before the Society of Chemical Industry. *Journal of the Society of Chemical Industry*, vol. xii., p. 325 (1893).

Mr. Ivan Lewinstein, the Chairman, stated that the deficiency of light on Sundays in the Hulme district being as great as on week-days proved that the pollution of the atmosphere was to a large extent due to domestic fires.

Mr. Grimshaw said manufactories contributed about 20 per cent. of smoke-pollution in large towns.

Mr. Teny said the alternatives were to burn either gas, anthracite or ordinary coal in an improved grate. He thought the last was the only solution.

The hydrocarbons produced and voided through the chimneys were paraffines, not benzenes.

21. Rauchbeschädigung in dem von Tiele-Winckler'schen Forstreviere Myslowitz-Kattowitz, Insbesondere Ermittlung, Bewerthung und Vertheilung des Rauchschadens. Carl Reuss, Herzogl. Anhalt. Regier. und Forstrath zu Dessau. 4to, 236 pp., with two charts. J. Jäger u. Sohn, Goslar (1893).

Introduction.— . . . The present enormous use of coal and constantly increasing extension of chemical-works, furnaces, and other industries is commencing to expose injuries which cannot be overlooked. . . . "One can note how works hide their processes of manufacture, the kind and quantity of the materials employed, in order to render the recognition of the kind and amount of damage more difficult."

The precautions required by law to restrict the emission of injurious products in the air are evaded, etc.

On the other hand, gardens are purposely set out with expensive ornamental flowers, unsuited to the climate, and which have no chance of prospering, in order to profit by the damages for destruction by smoke.

The subject has reached an importance which justifies its consideration as a separate science.

Metallurgist's Report. By von Skal.—Queries : 1. What are the injurious components in the smoke? 2. To what works is the damage traceable? 3. In what proportion do they share responsibility for the damage?

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As to the injurious components of the smoke, the opinion of Dr. v. Schroeder, of Tharand, is authoritative. The metallic salts insoluble in water produce no injurious effect, and those soluble in water and arsenious acid only a very slight one. But the injurious constituents are SO_2 and HCl, and especially the first.

. . . The SO_2, which has been recognized as the commonest cause of the damage caused by chimney-smoke, is due to the sulphur in the coal and to the zinc- and lead-ores treated in the smelting-works. H_2SO_4 and HCl were observed only in the minutest traces in the chimney-gases, and not to be compared with the very large proportions of SO_2.

The percentage of S in the coals varies so much that v. Skal assumes, on Muck's authority (*Elementarbuch der Steinkohlenchemie*, p. 43), 1 per cent. in all the coals, which is often exceeded and seldom not reached.

Forestry Report. By Reuss.—Reuss agrees that the principal cause of damage is SO_2, but observes that he has frequently had occasion to note in compost manufactories very evident signs of injury through HFl, and in this connection gives the method of Herrn Schumacher for the determination of HFl in vegetation.

25 g. of the dried and pulverized vegetable matter is treated with 5–7 g. K, Na carbonate and water in a nickel dish and evaporated to dryness.

This is then coked in a platinum dish and the residue leached out with hot water and filtered. The carbonized particles on the filter are dried and incinerated while the filtrate is being evaporated. The ash and salt residue are then treated with 5–7 g. more K, Na carbonate, thoroughly dried in a platinum dish and brought to quiet fusion over the blast-lamp, until the mass shows no bubbles. The melted mass is several times boiled with distilled water, and washed with boiling water, the filtrate treated with a little tincture of litmus, and then with dilute HNO_3 till it assumes a violet color; then evaporated in a platinum dish, at the end with repeated additions of Schafgot's solution, whereby the SiO_2 separates and is twice filtered and washed. After all the solutions have been united and again evaporated the remainder of the SiO_2 (after the expulsion of the $(NH_3)_2$ (CO_2) is precipitated by zinc oxide-ammonia, filtered, and the filtrate, now entirely free of SiO_2, precipitated by $CaCl_2$ solution and heated till all the CO_2 is expelled. The remaining precipitate is filtered, thoroughly washed, dried and heated to redness, then treated with excess of somewhat dilute acetic acid, evaporated on the steam-bath till every trace of vinegar odor has disappeared, dissolved in hot water, filtered, and the residue washed with hot water. It is then dried, and heated to redness in a platinum crucible and covered with excess of concentrated H_2SO_4 to which a drop or two of water had been added, after a tared glass plate has been placed as a cover on the crucible. The crucible is then heated for several hours over a small flame at some distance but finally close to the latter until no further action of the vapor on the glass plate is observed. The cooled and cleaned glass plate is then dried and after 15 minutes' cooling is again weighed over H_2SO_4.

The final etching of the glass plate by HFl and loss of weight is calculated as Fl in such manner that 2 parts by weight of loss in the glass are assumed as 5 parts of Fl, which relation was reached by experiments on pure $CaFl_2$. . . .

In this manner in 1888 and 1891 I demonstrated the damage to vegetation by the HFl escaping from the phosphate-works in Vienenburg, etc. . . .

The acute injury is apparent when the plants or their dead parts appear generally red to reddish brown. Little by little, through weathering, this color changes, depending upon the tenderness of the leaves, from a light or darker brown to black. Young leaves and tendrils that have been completely killed, crumple up and seem withered, and retain for a time a greenish color which only later is transformed to blackish.

When the injury is slight the leaves of the deciduous trees become dirty, pale;

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bleach, and remain sickly and small: take on too soon a weathered, dirty autumn coloring, and soon wither.

The evergreens behave similarly. At the commencement of the injury the needles, sometimes the older, become pale, strangely colored, dirty green, first on the upper side, which is more exposed to the smoke. Little by little all the needles become sickly, die and fall off, beginning with the older.

By continued smoking and increasing weakness of the tree the younger to the youngest plants die, and simple dry twigs and tops are observed. After a while individual trees are killed and bring about patches in the woods, which finally unite and produce barrens.

Even with this the smoke-injury is not done. The ground is made to share the injury in no small degree. Even at the beginning of the formation of gaps a growth of grass appears which rapidly consumes the provision of the humus. As usual in uncovered ground, the poorer berry-plants follow, and finally the heather, until by continued smoke-devastation even this disappears and the ground is the prey of wasting floods and dissipating winds.

With the appearance of chronic injury to the leaf there is generally noticed a blackish coloration of the bark, which is partly due directly to the smoke, and partly to the increased destruction and more rapid weathering of the outer epidermis. Also, an unusual collection of fallen undecomposed needles on the ground of evergreen forests may be taken as a tolerably sure sign of considerable injury by smoke.

It is gratifying for the corroboration of the visual test when this can be restricted to one kind of tree, since thereby the confusing differences due to the varying powers of resistance of different kinds of trees are avoided.

. . . Frequently alongside of perfectly sound or at least little injured trees are found very seriously affected growths and those nearly dead. The injury of a tract can never be denied because of individual sound trees found within it, but must be judged by the general condition of all the plants within it, with special attention to all observed injuries.

The degree of injury of a tract is not to be determined by the average of the observed injuries but by the most severely injured, which can be recognized with certainty.

As a tree for the investigation the fir was used because it was almost everywhere present, and owing to the needles remaining long in normal condition it is suited to make even lighter injuries apparent to the eye. For the rapid and exact definition of the injury of a tract the author grades the degrees of injury as follows:

	Injury.
Needles and stem healthy,	0
Older needles pale, dirty green, sickly,	1
Needles of older trees dead and fallen needles sparse,	2
Occasional twigs without needles, and dead from smoke,	3
Majority of twigs dry, the tree nearly dead,	4
Occasional trees dead from smoke,	5
Large number of trees dead; gaps in the tract,	6
Entire tract destroyed by smoke with exception of a few trees,	7
Ground vegetation nearly killed by smoke,	8
Ground vegetation entirely killed,	9
Land barren and wasted by water and wind,	10

In the tracts containing trees 40 or more years old the degree of injury is 3 and more. In young plantations the injury is less noticeable. . . .

Chemical Examination.—It is desirable to demonstrate by careful chemical analysis the kind and extent of the damage, and the fact that the whole region has been in-

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jured by the SO_2 with which it is infected. The examination was confined to the firs. In October, 1891, ten tests were made, distributed evenly over the entire district. The needles of the 1891 specimens were carefully separated from the older and both separately tested for H_2SO_4. In a table the older needles are shown to contain more H_2SO_4 than the younger, running from 0.53 to 0.87 in the older and from 0.44 to 0.74 in the younger.

The normal percentage of H_2SO_4 for fir-needles is 0.2 in the Harz and is assumed as the same in the Myslowitz–Kattowitz region, though it was not possible to find a specimen free from smoke-poison to establish the fact. 30 km. from Kattowitz the fir-needles from a place somewhat affected by locomotive smoke gave for the older needles 0.23 and for the younger 0.14; or 0.19 for an average, which confirms the results in the Harz.

In the Myslowitz–Kattowitz district the highest percentage of H_2SO_4 is four times, and the lowest twice, the normal.

The explanation of the less percentage of H_2SO_4 in young than in old needles is that the former have not been so long subjected to the action of smoke.

The development of the needles begins early in May and ends about the beginning of July. The average life of the young needles was about one month. In one month the needles take up 51 per cent., and in five months 71 per cent. of the H_2SO_4 content of the old needles.

(Tables and discussion of the method of taxing the damage.)

4. *Distribution of the Damage Among Particular Works.*—The task of the expert is to discover the extent to which each works is responsible for the damage.

The amount of SO_2 is not the only criterion of the amount of damage done. Two regions might be equally exposed to smoke and one suffer much more than the other. The height of the chimney-stacks is about 40 m. (131 ft.) In only two cases were the stacks 100 m. (328 ft.) high. It was expected that the higher chimneys would lessen the amount of damage by allowing the acid gases to become more dilute before they reached the vegetation. Experience teaches that these assumptions are only realized to a moderate degree.

In spite of the unquestionable benefits of the apparatus generally associated with stacks (dust chambers, etc.), and the fact that the nearby vegetation is not so much injured from a high as from a low stack, there is some damage done with high stacks to even the nearest vegetation, and remote vegetation is reached by smoke from high stacks which would not be injured otherwise.

22. Smoke-Abatement with Reference to Steam-Boiler Furnaces. Geo. Caruthers Thomson, F. C. S. *Proceedings of the Philosophical Society of Glasgow*, vol. xxvi., p. 148 (1894–95).

Bituminous coal was introduced as a fuel in the Thirteenth Century, but in 1306 a decree was passed forbidding its use. Many authorities are cited against the use of coal from this time on, including Count Rumford. The police and other regulations are given, with the percentages of punishment for their infraction.

Dr. Wm. Wallace, in a short paper read before this Society in 1880, shows that part of the sulphur is in pyrite and part in an organic compound. He estimates only half of the content to be volatile. . . . The area of the outlet at the top of the chimney should not be less than the area of the main flue. . . . Mr. Alf. E. Fletcher has shown that in some cases, when black smoke was emitted, CO was present, but when the chimney-top was clear no CO was emitted. R. Irvine estimates soot as 3 per cent. of all the smoke; Cohen and Hefford give 5 per cent. . . . According to G. Gruner, 1892, the fires of Dresden deposit about 4,800 cu. m., or nearly 1,000 tons, of soot, equal to 20 kg. of soot daily on each sq. km., 0.69 grain per sq. yd. . . . Mr. J. Aitken, F.R.S., says: "If we could get a fuel

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without sulphur we should get rid of a powerful element in city fogs." The sulphur compounds rather than soot are the cause of fogs.

Siegfried Hamburger, in a paper on Injury to Vegetation, etc. (*Journal of the Society of Chemical Industry*, vol. iii., pp. 203, 343 (1884)), quotes from Dr. Angus Smith substantially as follows :

In a part of London where coal is in the main only used for domestic purposes I found 730 grains SO_3 in 1,000,000 cu. ft. of air. In Manchester 1,098 grains.

Hamburger found in St. Helens 1,260 grains. Fletcher (1879) calculates that the gases escaping into the air at St. Helens per week contain :

Fire-gases, ...	800 tons SO_2
Copper-works, ...	380 tons SO_2
Glass-works, ...	180 tons SO_2
Alkali-works, ...	25 tons HCl

The inspectors sometimes find a larger quantity of sulphuric anhydride escapes from coal-combustion than is allowed to escape from sulphuric acid works.

23. THE EXTINCTIVE ATMOSPHERES PRODUCED BY FLAMES. Prof. Clowes, D.Sc., and M. E. Feilmann, B.Sc., *Journal of the Society of Chemical Industry*, vol. xiv., p. 345 (1895).

This paper treats of careful experiments to determine the composition of the atmospheres which cease to support flames of various combustibles.

24. UEBER DIE BESTIMMUNG VON SCHWEFLIGER SÄURE UND SCHWEFELSÄURE IN DEN VERBRENNUNGSPRODUKTEN DES LEUCHTGASES. Uno Callan, *Zeitschrift für analytische Chemie*, vol. xxxiv., p. 148 (1895).

Proves that the greater part of the sulphur on burning lighting-gas is transformed into SO_2, and 93.3 per cent. of this SO_2 was changed in the absorption-liquid to H_2SO_4.

25. THE ALLEGED ESCAPE OF CARBONIC OXIDE AND UNCONSUMED CARBON FROM COAL-GAS FLAME. Lewis T. Wright, *Journal of Gas Lighting*, vol. lxvi., p. 1023 (1895).

The writer concludes from a number of tests that there is no escape of CO or of unconsumed gases from the gas-flames of Auer (Welsbach) or other gas lights.

25A.—AN EXAMINATION OF THE ATMOSPHERE OF A LARGE MANUFACTURING CITY. Prof. Charles F. Mabery, *Journal of the American Chemical Society*, vol. xvii., p. 105 (1895).

26. RAUCHBESCHÄDIGUNG IN DEM GRÄFLICH V. TIELE-WINCKLER'SCHEN FORSTREVIERE MYSLOWITZ-KATTOWITZ. NACHTRAG ZU DEM WERKE GLEICHER BEZEICHNUNG V. JAHRE 1893 UND ENTGEGNUNG AUF DIE SCHRIFT, "WALDSCHADEN IM OBERSCHLESISCHEN INDUSTRIEBEZIRK, EINE RECHTFERTIGUNG DER INDUSTRIE GEGEN FOLGENSCHWERE FALSCHE ANSCHULDIGUNGEN," VON PROFESSOR DR. B. BORGGREVE; SOWIE WIDERLEGUNG EINIGER VON ANDERER SEITE GEGEN MEIN WERK "RAUCHBESCHÄDIGUNG, ETC." (1893) ERHOBENEN EINWÄNDE, MIT EINER KARTE. Carl Reuss, Herzöglich Anhaltischem Oberforstrath. J. Jäger und Sohn, Goslar (1896).

This is a bitter polemic directed at Herrn Borggreve's attack on his former work. Accompanying it is a map of a territory 60 km. E. and W., and 50 km. N. and S., of a part of Upper Silesia which is interesting as showing Reuss's results in hundredths of a per cent. of H_2SO_4 for the vegetations of all parts. The determinations vary from 0.21 per cent. in regions where there are few industrial works (*i. e.*, between Kobier and Mezerzitz), to 80 in the vicinity of the most densely occupied industrial district (between Myslowitz and Kattowitz).

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At a distance of 200 m. (656 ft.), the advantage of the high stack ceases to be apparent. In damp, still, heavy air the smoke with its SO_2, which is heavier than air, falls rapidly to the ground. When the wind blows continually one may observe the smoke escaping from the high stack gradually sinking and holding together for considerable distances; so that it does not suffer any unusual dilution. If one observe the smoke poured out from chimneys of different heights, at first that of each chimney can be separated, but in from 100 to 1,000 m. (depending upon the differences in height) the different masses behave as if they exerted an attraction on each other and soon mingle into a thick cloud which pursues its further course as a single mass. From these facts the difference in height of 60 m. (197 ft.) does not produce a measurable difference in the amount of injury.

The amount of SO_2 which reaches the vegetation will depend (1) on the amount of acid which escapes from the chimneys; (2) on the distance of the latter from the vegetation; (3) on the direction of the wind.

The injury is increased by dampness and lessened by dryness.

The vegetation can stand a certain amount of acid gases without injury. Beyond this amount it suffers. (Tables, and a map of 3,000 sq. km. of Upper Silesia, with figures indicating the number of hundredths per cent. H_2SO_4 found in the vegetation of the different parts.)

27. Notes on Poisoning by Carbonic Oxide. Douglas Herman, *Journal of the Society of Chemical Industry*, vol. xv., p. 854 (1896).

The author found mice, cat and rats died in a stable 50 yd. away from produce-plant. Communication only through loose soil. Mice are very susceptible to CO. With man at rest it takes about 20 times as long for a man as for a mouse to be affected. The first indication of poisoning is a dizzy, drunken feeling. Cold at extremities. Action of CO and alcohol similar in withdrawing O from the blood, but whereas alcohol stimulates the heart and provides to a certain degree the antidote CO does not. Affinity of CO for hæmoglobin is strong—250 times greater than that of oxygen.

28. The Combustion of Coal and Gas in House Fires. J. B. Cohen and G. H. Russell, *Journal of the Society of Chemical Industry*, vol. xv., p. 86 (1896).

The average percentage of soot from eight good Yorkshire, two Durham, and two Wigan coals amounted to 6.5 per cent. of the carbon consumed.

The rest of this elaborate paper concerns the evolution of CO, CO_2, and the heating effects; the comparative cost of gas and of coal fires.

29. The Determination of Sulphurous and Sulphuric Acids in the Products of Combustion of Illuminating-Gas. M. Dennstedt and C. Ahrens, *Zeitschrift für analytische Chemie*, vol. xxxv., p. 1 (1896).

By a series of elaborate experiments the authors assert the error of Uno Collan that the greater part of the sulphur in the ordinary as well as in the non-luminant gas-flame burns to SO_2. Their tables give 88.27 and 81.90 per cent. SO_2 in a very luminous flame and 62.55 in a blue flame; to 10.73, 18.20 and 37.45 per cent. SO_3, respectively.

30. A Method of Estimating the Weight of Solid Matter in the Air. J. B. Cohen, Ph.D., *Journal of the Society of Chemical Industry*, vol. xvi., p. 411 (1897).

In an experiment at Leeds the author roughly appraised the amount at about 1 mg. in 100 cu. feet.

The first method, by aspirating the air, using a Beckwith fan, and collecting dust on a glass plate smeared with glycerine, was found unsatisfactory.

The second method was by filtering a small and more carefully measured volume of air, and weighing the solid matter as before. The author employed two bags

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with an open zig-zag tube coated inside with vaseline. Experiments were made to determine if the passage of the air-current caused the vaseline to lose weight.

The result of five hours' aspiration of cotton-filtered air showed a loss at ordinary temperature of 1 mg. ; and in another experiment of six hours, 0.2 mg.

With deduction of this loss 100 cu. ft. of air was found to contain 1.36 mg. solid matter. In Leeds the average is 1.2 mg. per 100 cu. ft.

31. VERUNREINIGUNG DER LUFT IN DEN REINIGUNGS- UND REGENERIRHÄUSERN DER GASANSTALTEN. H. Drehschmidt, *Journal für Gasbeleuchtung und Wasserversorgung*, vol. xl., p. 517 (1897).

It has been often asserted that the workmen in cleaning- and regenerating-works have been injured by inhaling the air in such places, but the spectroscopic examination of their blood has not confirmed this supposition. The police of Berlin required investigations to be made. Dr. Hans Wolf, Assistant at the Gas Works Laboratory, has done most of the work.

The poisonous ingredients are ammonia, hydrocyanic acid, hydrogen sulphide, carbon disulphide, and other sulphur compounds of carbon. Ammonia is eliminated by the dry-cleaning process to 1 cc. in 100 cu. m. The other gases and vapors, excepting the carbon oxides, are almost completely withdrawn by the dry-cleaning. The hydrated iron of the cleaning-apparatus might hold the noxious gases so feebly that they would be disengaged on taking the material out, but not H_2S, etc.

CO_2 can only appear on the emptying of the cleanser.

Hydrocyanic Acid.—The cleaned substance, if subjected only once or twice to the process of the cleanser and spread out on the regenerator floor, contained 0.000066 and 0.00014 vol. per cent. HCy. After having been used 13 or 14 times 0.00012 and 0.00012 vol. per cent. HCy. As more HCy was proved to exist in the older masses, the amount in the air was determined when the charge had lain a considerable time on the floor of the regenerator. 0.00002 vol. per cent. HCy was obtained. No case has been found where the breathing of the HCy in the air has proved injurious. The author does not believe any injury from this cause has occurred to workmen.

Ammonia.—400 to 600 liters of air were passed through normal H_2SO_4 for the test. When the mass had lain some time on the floor of the regenerator the air contained 0.00002 vol. per cent. NH_3.

When first brought to the regenerator, 0.00004 vol. per cent.

On emptying the cleaning-boxes, 0.00041 vol. per cent.

Hydrogen Sulphide Gas.—A qualitative test with moist lead-acetate paper gave hardly visible indications of this poison.

Carbon Monoxide.—This was collected in the cleaning-vessels.

The emptying lasted two hours. The samples of air were passed through potassium solution and then through red-hot platinum capillaries, and the resulting CO_2 determined by baryta-water. In the samples examined were found 0.006, 0.027, 0.038, 0.032 vol. per cent. of CO.

No symptoms of poisoning appear (according to W. Hempel) unless the CO reaches 0.043 per cent. of the volume of air containing it.

The results prove that the work in the cleaning- and regenerating-houses, under ordinary circumstances and with familiar precautions, is not injurious to those undertaking it.

32. VEGETATIONSSCHÄDEN DURCH GASAUSSTRÖMUNG. Dr. Otto Pfeiffer, *Journal für Gasbeleuchtung und Wasserversorgung*, vol. xli., p. 137 (1898).

It is well known that the trees of large cities are injured or killed by gas. The lindens of the street in Berlin to which they give the name are an example.

In other cities the same state of things exists. It was natural that the leakage from the network of illuminating-gas pipes was first blamed. This has also been proved

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by a number of experiments. The most important experiments were made by the Berlin College of Magistrates in 1871 in the botanical gardens and also in private gardens, according to the report of Virchow. The subjects of experiment were maple, linden, plantain, silver-poplar, acacia, etc., against the roots of which during several months to a year measured quantities of illuminating-gas were constantly led. By these and similar experiments (especially by those of V. Böhm) the poison symptoms were determined, beginning with the withering and falling of the leaves, while the rootlets were completely decayed and killed; and Kny observed a curious bluish color in sections of roots with diameters up to the thickness of a finger, increasing in strength from the center to the periphery: which indicated that the poison entered the growing roots with the nutritive material, and not through the bark of the older roots. From this the poison from illuminating-gas must be accepted as a fact.

Freytag thought clean gas free of tarry products was not injurious to vegetation. Later, Poselger concluded that illuminating-gas had no influence on plants.

The author experimented with a gas-pipe 3 m. long and 100 mm. in diameter, filled with sandy garden soil. For ten days 2.4 cu. m. illuminating-gas were daily conducted through this earth and consumed in a burner at the end. It was noticeable that the gas lost much of its characteristic odor by passing through the earth. When saturated, the earth was taken out and aërated, a part being transferred to the laboratory to be examined for the characteristic components of illuminating-gas. No ammonia or cyanogen was found.

Some results were obtained in proving the presence of phenol-like bodies, but they were not completely satisfactory.

The proof of heavy carbon sulphides was more satisfactory. 500 g. of earth was left several hours in a flask with 750 cc. of water and occasionally shaken. This was filtered as many times as was necessary to get a clear solution, which had no characteristic peculiarity but an earthy smell. But if one add H_2SO_4 and then a few cc. of bromine-water, this disappears if carbo-sulphides be present. Dr. Pfeiffer puts 100 cc. of the filtrate in an Erlenmeyer matrass, acidifies and allows so much of a dilute bromine solution (10 parts saturated Br solution diluted to 100 parts) to flow out of a burette with glass cock until a yellow color is visible which does not immediately disappear. By a blind test with distilled water it will be found that 0.5 cc. Br water can be detected in the liquid. 100 cc. water after having leached earth which had been impregnated by gas requires 10 per cent. Br water in cu. centimeters.

(*a*) Earth exposed to the air after a gas absorption of several hours, 9 to 10 cc.

(*b*) After nine days kept in a sack, 9.7 to 10.6 cc.

(*c*) Nine days kept in a sack and afterwards dried four days in a thin layer at the temperature of a living-room, 10 to 13.3 cc.

(*d*) 100 days kept in a sack, and after the loss of all moisture of the soil, 11.7 to 12.3 cc.

(*e*) The same after 111 days, 12 cc.

(*f*) The same after 164 days, 11.6 cc. Br water.

From all of this it appears that the strength of the reaction does not diminish after lying half a year. As a practical test he cites the case of breakage of a pipe, in the immediate neighborhood of which a test was taken from a soil unfavorable to absorption. After lying eight days in the laboratory a leaching of 100 cc. of the earth required 5 to 6 cc. Br water.

From the above, one is justified in concluding that a gas-pipe has broken when an investigation of the earth gives the tests for heavy carbo-sulphides, which can be established by the Diazo reaction, but the assumption must be excluded if the reaction cannot be obtained.

If the gas-works would permit specimens of the earth to be examined in places

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where leakage is suspected it would more and more diminish the prejudice against illuminating-gas.

33. Colorimetrische Rauchdichtebestimmung. P. Fritzche, *Zeitschrift für analytische Chemie*, vol. xxxvii., p. 92 (1898).

To determine the density of the chimney-smoke a measured volume is drawn into a suitable glass tube furnished with asbestos, and the carbon estimated by heating the tube in oxygen or air and determining the CO_2. This is time-wasting, and besides is only adapted to a laboratory.

A glass tube of about 10 mm. interior diameter and 150 mm. in length is filled with 2 g. of loose cellulose (nitric cellulose). By a short piece of rubber-tubing this glass tube is connected with another glass tube equal in diameter, of which the end extends within the interior of the chimney or duct. The other end of the apparatus is connected with an aspirator which draws in from 10 to 20 liters of furnace-gas through the cellulose.

At the conclusion of the experiment the apparatus is taken apart. The uppermost black cellulose layer is removed by a pincette to a wide-mouthed stoppered flask of about 300 cc. Together with the partly colored remaining cellulose, both glass tubes are washed out so that the entire soot comes into the cellulose, which is then transferred to the stoppered flask, covered with 200 cc. of water, and well shaken, so that a uniform gray-colored liquid results. In order to judge the amount of soot from the color of the fluid, it is poured into a test-tube 40 to 50 mm. in diameter, with round bottom, and the color compared with a color-scale prepared beforehand.

The scale is prepared by placing 3 g. cellulose in each of a number of tubes containing 5, 10, 15, 20, 25 and 30 mg. soot, to which 200 cc. water has been added and the tubes shaken.

34. Zur Beurtheilung von Rauchschäden. H. Ost und C. Wehmer, *Die Chemische Industrie*, vol. xxii., p. 233 (1899).

In 1893–94 the authors undertook an investigation of the cause of certain spots, especially on the leaves of mayflowers and roses, which resembled the effects of acid chimney-gases, but were neither due to these nor to parasites, insects, frost, wind nor dry-rot. They had previously said in print: "On the leaves of various roses, even of those well cared for, often appear violet spots, perhaps due to insect-stings. In the middle of the violet-colored living cells there often appears a sharply defined rust-red spot, the result of death, which resembles an acid-spot to the point of mistaking one for the other. . . . With Dr. Wehmer these spots were examined under the microscope, but no point of difference between the two kinds was discovered.". . . Studies were pursued, principally with roses from gardens and beds northwest of Hanover and practically free from acid-gas. A very common malady is seen in the light or dark violet spots which appear and often cover large areas in the living tissue on the upper side of the leaf, on the edge, and in the middle. The authors observed these spots every year in all collections of roses within and without the city and on the most various kinds, such as Centifolia, La France and Malmaison; they were most strongly marked in spring and autumn, but also found in midsummer when it was wet and cool.

They are probably results of cold and wet weather, as examples from 1896 and 1898 prove. By marks made on leaves in 1898 it was discovered that the violet spots changed again to a duller green. As the respective cells are living, this change of color is not astonishing. More violet is found at the commencement of October-November frosts, long before the leaf exhibits any portions killed by frost; the upper side reddens more and more, and the reddening at last attacks the under side. These violet spots in living tissue are not to be confounded with acid-spots. But the

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brown and rust-red spots are deceptively similar to the latter. [Here is introduced a beautifully colored plate illustrating the spots on various rose-leaves.—ED.]

After describing the various spots illustrated in the colored plate the article proceeds: The violet color is probably identical with the often-described but not thoroughly known anthocyanogen.

In dried leaves it is not altered by months of exposure to light, while chlorophyll disappears. Alkalies color it bluish green; acids reddish violet; dilute SO_2 bleaches it slowly; sulphuric acid or extended action of air restores the color, as in the case of the red color of rose-leaves.

Experiments on a number of potted roses in 1898 showed that when treated with SO_2 and afterwards exposed to the sunlight and air, the dead spots, extending through the entire twig, at first discolored, in two or three weeks turned brown or red, and consisted of air-filled collapsed cells with brownish plasma residues. The entire absence of the violet ring in all hitherto examined sulphurous-acid spots is worthy of notice. The dead spots are always surrounded by a narrow blackish zone, in sharp contrast to the green tissue, and this dark boundary proves under the microscope to be free from violet.

Earlier observers have noticed this dark band—viz., Schröder, and Reuss and Hasenclever—but as a characteristic indication of acid- or smoke-injury it has not yet been mentioned, nor more closely investigated.

Nor do the authors wish to generalize too much from the small number of spots they have examined; but they think themselves justified in saying that, at least on rose-leaves, an acid- or smoke-spot is to be recognized by its dark-colored zone. In spots from all other causes, in their examinations, this appearance has been wanting.

Even if one may exclude fungus, insects, wind, frost, and other known causes of spots, in their judgment one is not justified in assuming acid or smoke to be the cause without direct and positive proof.

The article concludes by a description of the gardens and alleys NW. of Hanover, where the observations were made. Although to the south there are large industrial works, cotton-spinning, ultramarine manufactories (the latter of which has been compelled during the last four years to eliminate the acid from the products of combustion, and has the highest chimney, 60 m., or 197 ft.), there is no proof of injury to the vegetation by furnace-gases, although the city's yearly consumption of coal is 450,000 tons.

In the opinion of the authors, the injury due to chimney-gases has been exaggerated. Sometimes in the early morning, and with a southerly wind and heavy atmosphere, the SO_2 from the manufactories is noticeable and unpleasant in the hawthorn alley near the High School, yet the vegetation does not seem to have been injured, although the hawthorn is considered especially sensitive to smoke.

35. SMOKE AND ITS DIMINUTION. B. Donkin, *Engineer*, vol. lxxxvii., pp. 507, 637 (1899).

Increase due to exhaustion of wood and use of coal. Source, factory-boiler and domestic fires. Assuming 5½ million inhabitants of London it is fair to assume that two million domestic chimneys smoke between 7 a.m. and 11 p.m. during winter, and half a million in summer. From 7 to 9 in the morning, when fires are lighted, most smoke is given off. It would be reasonable to assume the amount of smoke given off as equal to that in a chimney 1,000 ft. square.

Restaurants are the greatest offenders, and their number is increasing inordinately.

Nature of smoke. Result of chemical decomposition of coal with insufficient air for complete combustion. Its unburned tarry constituents make up the principal part of smoke. Smoke should be abolished for sanitary reasons.

D. K. Clark, in his *Smoke Abatement* gives ten small pictures of shades of

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smoke. Density of smoke has been determined by drawing chimney-gases into previously weighed wool. Prof. Lewicki, of the Saxon Smoke Commission, in 1896, determined the soot in the gases of combustion. Methods are not entirely satisfactory to the author.

In Minay's process the gases are passed through asbestos.

Soot is dried in a current of air and burnt, and the CO_2 determined. The quantity of soot per cu. meter is measured. Ingenious method proposed by Dr. Fritzsch in *Zeitschrift des Vereines deutscher Ingenieure.* The combustion-gases are drawn off and deposited in cellulose, mixed with water and estimated by a color scale. (See *ante.*)

The author introduces a piece of cardboard, 1 ft. square, coated with some adhesive substance, into the bottom of the chimney, and judges the nature of the smoke by the quantity and character of the particles which adhere.

The South Kensington, Manchester, Smoke Abatement Commission used scales of ten shades. The author does not indorse this.

The second English Smoke Commission adopted a scale of only three shades—*i. e.*, "faint," "medium" and "black." Author thinks ten shades too many, and three too few. The best scale (Ringelmann's) is of five shades, and is in use in Switzerland: (1) white transparent vapor; (2) light brown smoke; (3) brownish gray smoke; (4) dense smoke; (5) thick black smoke.

Ringelmann's plan is to represent the different grays into which the shades of smoke are divided by black cross-lines on white paper.

At a certain distance they represent the desired smoke-tints. They are hung up so that the observer can see them and at the same time the smoke from the chimney. These diagrams have been printed in France and the United States.

No. 0. No smoke—All white.

No. 1. Light gray smoke. Black lines 1 mm. thick and white spaces of 9 mm. between.

No. 2. Darker gray smoke. Black lines 2.3 mm. thick and 7.7 mm. apart.

No. 3. Very dark gray smoke. Black lines 3.7 mm. thick and 6.3 mm. apart.

No. 4. Black smoke. Black lines 5.5 mm. thick and 4.5 mm. apart.

No. 5. Very black smoke—All black.

Reischle, chief engineer of the Bavarian Boiler Association, has laid down certain rules for combustion. The HC's must be brought rapidly to a very high temperature before they are allowed to escape; and must be supplied with sufficient air so admitted as to thoroughly mix with the gases (boiler-grates).

On the Continent authorities agree that air should be admitted in two places: in front of the fire, and at the back near the fire-bridge. Mr. Spence, of New Castle, experimented with admitting air above and below the fire-bridge until finally the smoke disappeared.

Headley, in America, admitted the air through hollow passages—brick-work of flues; another plan was through hollow fire-bars.

Careless stoking is the cause of much evil. In boiler-furnaces two methods are good: (1) The American down-draft with two grates. Author thinks this too complicated; does not indorse it. (2) Powdered-coal firing. Author hopeful of this.

Prof. Lewis's four methods of preventing smoke—(See *ante*)—viz., use of anthracite; consumption of products; gaseous fuel; and condensation of tarry components of smoke—all have objections.

English Smoke Commissions.—1881, and Manchester, London, etc., in 1895. Branch Commission, Sheffield, recommend restriction of emission of black smoke from a boiler-chimney to two minutes per hour for one boiler and three minutes for two boilers. All commissions agreed that domestic fires are the most pernicious.

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German Smoke Commission.—In Berlin, 1894, a commission was appointed to test various kinds of grates.

Paris Commission.—1894. Observations were made by two persons, each pressing a pen on a moving coated drum graduated by lines to record the different shades of the five-color scale, and also to record time in minutes. The prize was given to the English mechanical stoker. The apparatus which showed the least smoke, did not, however, give the highest efficiency in evaporation.

36. Address of President George Beilby of the Society of Chemical Industry. *Journal of the Society of Chemical Industry*, vol. xviii., p. 643 (1899).

Output of coal in the United Kingdom, 202 million tons. Smoke-nuisance treated from two points of view: (1) scientific investigations of chimney-smoke; (2) various remedies applied. Root of smoke-evil the raw coal burned, and full fruition insured by the method of burning. Total coal consumed during 1898 in the United Kingdom, 157 million tons, of which 76 for power; 81 for heat (46 industrial and 35 domestic). For power, railways, 10 to 12; coasting-steamers, 6 to 8; mines, 10 to 11; factories, 38 to 40. For heat, blast-furnaces, 16 to 18; steel- and iron-works, 10 to 12; other metallurgy, 1 to 2; chemical, pottery, glass, etc., 4 to 6; gas, 13 to 14.

From observations of the exhaust of locomotives, the author thinks steam with the smoke causes rapid deposit of soot. Vegetation along the lines injured, also by steam-boats in narrow rivers, but of course not in coastwise trade. In factories, etc., the classification ought to be (1) hopelessly smoky, and (2) potentially smokeless. The contractors' vertical boiler, the egg-ended boiler of the small city factory, the multitubular boiler for electric-lighting, derived from extinct threshing-machines, are of the first class; the Lancashire boiler of the second. Generation of electricity by steam produces dense black smoke; an anomaly in an apparatus designed to insure purity of light and air. In 1899 the Glasgow and West of Scotland Smoke Abatement Association issued a report on firing Lancashire boilers by hand, and by mechanical stokers. The conclusion was, that means were now known to enable one to work boilers without smoke. In 1898 the Manchester Committee for Testing Smoke-Prevention Apparatus (the outcome of a suggestion made by the Chief Inspector under the Alkali Acts, etc.) concluded that a manufacturing district may be freed of smoke (at least from steam-boilers) by carrying out the suggestions in their report.

Remedies: First, mechanical aids to combustion; second, manufacture of smokeless fuels.

Of the first class, mechanical stokers, which may be divided into coking and sprinkling.

In the first, the coal is passed in at the front of the furnace, where the gases are given off and pass over the glowing coals.

Sprinkling stokers distribute the fresh coal over the whole surface of the furnace, but without chilling the fire.

The limits of "throughput" ["Durchsatz"—Ed.] have been widened, so that the throughput may be dropped from 100 to 25 without interfering with economy and smokelessness. Various distributions of air and fuel belong in the class. The author's own experience is that with care and skill smoke can be reduced to an absolute minimum.

Of the second class, destructive distillation in one of two ways—*i. e.*, gas-retort and coke-oven.

(Here follows a description of the details of the working of these various methods, with values of the by-products.)

With a raw-coal value of 1 the various products may vary in value from 1.5 to 3.

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Liquid fuel will have a value of 1.5 to 2.5; gas (of 600 B.t.u. per cu. ft.) a value of 2 to 3; coke and briquettes a value of 1 to 1.5.

Coke is not popular as a fuel and is not adaptable to miscellaneous uses for fuel. The remedy is the briquette made with it and tar.

The Partial Combustion of Coal for the Production of Fuel Gas.—There are two producers: the continuous and the intermittent. The continuous gives gas of uniform quality by burning the fuel with a limited supply of air, or air and steam. The intermittent gives a gas rich in combustible components. Air is forced through the mass to heat it and then stopped and steam supplied, producing a rich gas of 350 to 370 B.t.u. First class, 130 to 160 B.t.u.

Remedies for modern methods of heat production.

(Here follows a description with illustrations of the method of introducing gas-heating into private houses.)

37. RECHERCHES SUR LA FORMATION DE L'ACIDE AZOTIQUE PENDANT LES COMBUSTIONS. M. Berthelot, *Comptes Rendus, Academie des Sciences*, vol. CXXX., p. 1345 (1900).

Formation of oxides of nitrogen during combustion of carbon and hydrocarbons had been observed by Cavendish, but not undertaken systematically. In more than a thousand determinations of heat of combustion and formation of organic compounds the author had to determine each time the minute quantity of N_2O_5 formed by the nitrogen contained in the oxygen employed in the experiments.

C, and the binary, ternary, and quaternary compounds formed by the association of H, Cl, and S, with C, were the objects of this study. First C, S, and H.

The N_2O_5 was recovered either in water or in a dilute solution of KHO.

The mean of six experiments in burning amorphous C in an atmosphere of O containing 8 per cent. N was:

Amorphous carbon, . .	HNO_3 0.051 = 0.011 N per 1 g. C burned.
Graphite, ...	HNO_3 0.010
Diamond, ...	HNO_3 0.015
Rough diamond, . . .	HNO_3 0.017

Some ammonia was also determined, ± 0.00046, in amorphous carbon.

Second series, central combustion in an atmosphere of oxygen containing 8 per cent. N at constant atmospheric pressure.

Charcoal was heated to redness and transferred in a small capsule to a vessel containing O.

For 1 g. of burnt charcoal 0.00087 g. HNO_2 = 0.00019 g. N.

In round numbers the weight of the N is about 0.070 of that of the O united with the C; and the weight of O combined with the N is $\frac{1}{500}$ of that combined with the C.

Combustion of charcoal (amorphous C) in air under normal constant pressure: For one g. of burnt charcoal 0.000096 g. HNO_3 = 0.000021 g. N.

Let us suppose that, in the Department of the Seine, there are burned annually 4 million tons of combustibles of all kinds, coal, oils, etc. (which is the fact, according to the statistics); and assume that the conditions are like those of carbon in the previous experiments. There would result annually 367,000 kg. of HNO_3: say 1,000 kg. per diem. That would make for each hectare of the department 8 kg. derived from human industries. Taking all France, there would result from a similar estimate from human industries 0.1 g. per hectare. This is, however, much too low.

38. THE GREAT SMOKE-CLOUD OF THE NORTH OF ENGLAND AND ITS INFLUENCE ON PLANTS. Albert Wilson, *Report of the British Association for the Advancement of Science*, Section K—Botany, p. 930 (1900).

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The widespread effect of smoke insufficiently realized. Dwellers in towns often so hardened to it as to be almost oblivious to its presence. The great smoke-producing district of the North of England; its extent; miserable condition of vegetation in some parts of the area. Variation in amount of smoke according to the season Effect in reducing air-transparency; dimness of sky and landscape. Distance to which smoke travels. Smoke often mistaken for haze. Red sunsets in southeast Yorkshire. Atmosphere of the North of England. North of the smoke-area never brilliant with southerly winds. The smoke from Barrow-in-Furness, an isolated town; great distance at which this is noticeable; comparison of its volume with that from the great smoke-area. The characteristic smell from certain large works, and the distance at which it can be detected. Discoloration of rainwater; "black rain." Influence of smoke on sunshine and air-temperature in calm summer weather, and in anti-cyclonic weather during autumn and winter; low day-temperature maxima. Smoke and fog-production. Long-continued smoke-fog of February, 1891. Darkness in and around large cities. Effect of smoke on mosses and hepatics as compared with that on plants of higher order. Smoke at a maximum in winter, when many mosses are in a vegetative condition. Great diminution in their abundance and luxuriousness in the neighborhood of large towns. Peculiar exposure of bark-loving species to smoke-influence, and the cause. Threatened extinction of *Ulota* and *Orthotricha*.

39. Mineral Constituents of Dust and Soot from Various Sources. W. N. Hartley, F.R.S., and Hugh Ramage, A.R.C. So. I., *Proceedings Royal Society, London*, vol. lxviii., p. 97 (1901).

Nordenskjöld described two kinds of dust collected by him from Arctic ice: (1) diatomaceæ, and (2) felspathic sand. The third was probably from interplanetary space.

Prof. O'Reilly gave to authors: I. Solid matter which was carried down with hail and collected at Stephen's Green, Dublin; II. Solid matter carried by hail and sleet onto the window-sill of the Royal College of Science, Dublin; III. Pumice from Krakatoa.

I. Contained Fe, Na, Pb, Cu, Ag, Ca, K, Ni, Mn (Ga and Co?).

II. Fe, Ca, Na, Pb, Cu, K, Mn, Ni, Ag, Th (Ga, Ru?).

III. Fe, Cu, Ag, Na, Ni, K, Rb, Mn, Ga, In, Sr.

With the exceptions of Sr, Ni, and Co, the authors found the same constituents in 97 irons, ores and associated minerals. In six meteoric irons they have found the same constituents, with Ni and Co, the latter invariably in smaller quantity than the former.

(Tables of spectroscopic observations are given and explained.)

The authors present two conclusions:

1. The presence of Ni is not certain evidence of extra-terrestrial origin.

2. The dust which fell on calm nights, Nov. 16 and 17, 1897, was very probably cosmic.

The authors call attention to the distribution of Ga. (All minerals, flue-dust, soot, air-dust, iron-ores, bauxite.) They hope to find it concentrated in some mineral as are Th, Cs, Ge and In.

40. Some Observations on the Factory and Workshops Act and the Alkali, etc., Works Regulation Bill of 1901. Eustace Carey, *Journal of the Society of Chemical Industry*, vol. xxi., p. 214 (1902).

This is principally a discussion of the Acts of Parliament mentioned, with comments upon the intent of the phraseology, and the changes from the old Acts.

41. The Soot Deposited on Manchester Snow. Wilfrid Irwin, *Journal of the Society of Chemical Industry*, vol. xxi., p. 533 (1902).

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After a fall of snow in February, 1902, the author collected a layer 1 in. thick over 100 sq. in. of his garden, 3 miles north of the Town Hall in Manchester. Transferred to a dish, melted, filtered, and extracted solid matter. The dried residue was extracted with benzene, dried, weighed again, and ignited. The following are the results:

100 sq. in. contained 0.073 g. soot; therefore, one acre contained 4.58 kg. = 10.7 lb.; one sq. mile contained 3 tons 1 cwt.

Dr. Knecht, of the Manchester Technical School, took a sample in Whitworth Street (center of the town), and obtained three times as much soot. The difference was due to air-currents.

Almost 300 tons, or 30 tons per day, of soot must have fallen from the Manchester chimneys during the fall of snow.

To ascertain if the soot in the falling snow was a notable part of that observed to fall on the snow a sample of snow was taken underneath the top layer. Tested as before the soot contained:

	Per Cent.
Solid carbon with a little solid matter,	48.6
Grease,	6.9
Ash,	44.5

The percentage of ash was higher than was expected, and much higher than in the soot from chimneys.

The grease, or heavy oil, on heating smelt like burning wood. Though small in amount, it assisted the snow to adhere to the side of the vessel. The author thinks it plays an important part in causing soot to adhere.

Dr. Knecht's soot on analysis gave:

	Per Cent.
Solid carbon with fibrous matter,	45.1
Grease, or heavy oil,	3.5
Ash,	55.4

Prof. E. Knecht collected a layer 0.5 in. thick on a sq. yd. opposite new School of Technology in Whithworth street.

Insoluble residue left after boiling weighed 3.8 g. and contained soot, fibrous matter, and other *débris*.

The filtrate, of brownish color and acid reaction, left a residue of 0.406 g. Residue extracted by hot water and in the aqueous solution, 0.0106 g. Ammonia was obtained chiefly as sulphate. The insoluble part was crystallized $CaSO_4$, a product always present in domestic soot.

Another sample from a garden in Crumpsall contained, besides free acid, ammonium sulphide and chloride. Ordinary chimney-soot contains as much as 15 per cent. ammonium sulphide, which gives it manurial value. The manurial value in the country round is not inconsiderable if enough lime is present, either naturally or artificially added, to neutralize the free sulphuric acid.

42. The Relation of Sulphur in Lighting-Gas to Air-Vitiation. J. S. Haldane, M.D., F.R.S., *Journal of Gas Lighting, Water Supply, etc.*, vol. lxxxiii., p. 564 (1903).

Air in which gas is burned is more oppressive than that to which a proportionate amount of CO_2 has been added. The cause is sulphur. The average English gas contains 0.46 g. per cu. m. Sulphur is present usually nine-tenths as H_2S, and one-tenth in other forms, such as CS_2. The S in gas is largely responsible for the injury to the bindings of books.

(Experiments with gas-combustion product in two rooms.)

The chief conclusions are:

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1. The unpleasantness of air in gas-lighted rooms is due to the presence of sulphur in the gas, and varies with the amount of sulphur.

2. Gas purified of CS_2 (by purifier of $CaSO_4$ or other means) is greatly superior hygienically to gas only purified from H_2S.

43. EXAMINATION OF THE ATMOSPHERE OF THE CENTRAL LONDON RAILWAY. Frank Clowes, D.Sc. (1903).

Owing to the absence of combustion in the locomotives, the impurities of the air of the stations were due to the respiration of the passengers and staff. In the tunnel the amount of CO_2 decreased from the Bank end toward Sheppard's Bush terminus.

Dr. F. W. Andrews thus summarizes the results of his bacteriological observations.

1. Micro-organisms present in the air of tunnel as compared with that outside, were as 13 to 10.

2. The number of micro-organisms was great in proportion to the concentration of human traffic: highest in railway carriages, platforms, and lifts.

3. The air in the railway tunnel does not compare unfavorably with that in inhabited rooms.

4. No pathogenic germs other than those commonly present as saprophytes on the normal body were detected in sufficient quantity to analyze.

5. The number of organisms capable of growing at the temperature of the human body was much greater in the air of the C. L. Ry., but this was due to non-pathogenic *Sarcinæ* and other species.

6. The number of micro-organisms in the air is generally proportional to the degree of chemical contamination—with exceptions.

7. The species in the railway tunnel and in the free air are the same.

The author recommends that no part of the railway air should contain more than twice the amount of CO_2 found in ordinary air; 8 vols. per 10,000 should be the maximum.

44. DIE BESCHÄDIGUNG DER VEGETATION DURCH RAUCH. HANDBUCH ZUR ERKENNUNG UND BEURTEILUNG VON RAUCHSCHÄDEN. Dr. E. Haselhoff, Vorsteher der landwirthschaftlichen Versuchsstation in Marburg, a. d. Lahn, und Dr. G. Lindau, Privat-Docent der Botanik und Kustos aus kgl. botanischer Museum zu Berlin. Bornträger Brothers, Leipsic (1903).

1. *Origin of Smoke.*—Though the visible clouds of smoke may be unpleasant and injurious to vegetation, the real injuries arise from the invisible products of combustion. . . . The amount of SO_2 produced by coal combustion is so little that it will not pay to recover it commercially.

In Lord Derby's Alkali Act, sulphuric acid manufactories were not allowed to discharge more than 5 per cent. of the produced HCl gas. This clause was later altered to forbid more than 0.464 g. of HCl in 1 cu. meter.

A case is known where the waste gases of a sulphuric-acid works in a narrow valley do not reach 4 g. to 1 cu. meter.

Chr. Drelle mentions that in the granting of concessions for new works in Prussia, it is exacted that SO_3 in the waste gases should not exceed 5 g. in a cu. meter.

2. *Signs of Smoke-Injury.*—The rings of growth are difficult to inspect, but the extreme ends of the leaves and stems offer an admirable means for microscopic examination. In forest blights the ends of the needles are discolored. Finally, the needle changes more or less to red. In deciduous leaves, spots on the broader surfaces of the leaves are also more or less red. The spots either appear between the middle rib and the two side ribs, or they surround the leaf. By the manner of formation HCl can be distinguished from H_2SO_4. In young grain or grass the tips become first red, then yellow and finally white.

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(A list of plants is here given in the order of their resistance to injury from SO_2 and HCl.)

The wind is the principal factor in spreading the poisonous gases.

D'Arcet suggested a map with concentric circles around the source of contamination. The diameters in the direction of the prevailing winds were made longer than the others. It is not practical.

A better suggestion was made by Reuss. He divides the circle into eight sectors, with the smoke origin in the center. The north sector is between NNW. and NNE.; the NE. sector between NNE. and ENE., etc. South winds traverse the north sector, SW. winds the NE. sector, etc. Each sector for a radius of 1,000 m. contains 39 hectares (ha); between 1,000 and 2,000 m., 118 ha; and for each additional 1,000 m. up to seven, 196, 275, 353, 432, and 511 ha respectively. Taking the prevalent direction of the wind one can calculate approximately how many tons of SO_2 annually are carried over a given space. The weather, the peculiarities of the ground, and the effect of high stacks must be taken into consideration.

. . . Wislicenus establishes his areas of damages by analysis.

3. In some cases the injury through atmospheric influence resembles smoke-injury. Observation of two or more growth-periods will usually enable one to distinguish the difference, as also in many cases the chemical analysis.

From lack of potash the leaves are discolored to yellowish brown, which becomes white. Lack of P_2O_5 shows itself by the dark green color. Lack of iron from pallor. Some insects living in the interior of the leaves produce yellow or red blotches. The distinction from smoke-poisoning can be discovered by transverse sections under the microscope, when mycelium strings will be observed between the cells. . . . In the controversy between Reuss and Borggreve the latter ascribed the injury of the Kattowitz-Myslowitz district to insects, and not to smoke. The example proves that insects will infest a district already injured by smoke.

4. Freytag says the effect of the destruction of the plants by acid resembles their decay in the autumn.

Sachs observes that in the decay of leaves in the fall first the chlorophyll and starch disappear from the assimilating cells. The solutions of these are conducted through the stem of the leaf to the main stem, where they remain as wood parenchyma, as a reserve, whence they are distributed as needed. During this circulation the leaf cells are filled with a colorless fluid. (Detailed account of the chemical changes, etc.)

5. *Proof of Smoke-Gases in Injury to Vegetation.*—Another way to prove injurious smoke gases is the examination of the air at the place of injury.

Braconnet and Simonin thus investigated the vicinity of Dieuze, near Nancy, in 1848.

Chemical-works producing salt, H_2SO_3, $CaCl_2$, HCl, H_2NO_3, tin salts, lime and sodium carbonate. In the direction of the wind one could detect at a distance of 1.25 miles the odor of SO_2, HCl, and coal-smoke. At distances of 200, 500, and 1,000 m. around the works litmus-papers were suspended, and glass plates moistened with milk of lime. In one or two nights all the test-papers in the path of the wind were reddened if it passed over the works, but not otherwise. The potash-solutions on the glass plates were only partly neutralized, but no chlorine could be detected. The dew shaken from the plants showed a neutral reaction, but a determinable content of Cl, as also of H_2SO_3, Ca, alkalies and organic substances.

The dew of places exposed to wind blowing over the works showed traces of $CaSO_4$ and NaCl, but no $CaCl_2$ or NH_2Cl. . . .

G. Witz (*Comptes Rendus*, vol. c., p. 1385 (1885)), in Rouen, hung up printing-paper charged with lead oxide and noted that it gradually became colorless.

Ost (*Chemiker-Zeitung*, vol. xx., p. 165 (1896)) investigated the gases of the

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following works: sulphuric acid, compost-works, ultramarine, chemicals, salt-works, fire-brick-works, and four others. He used, to collect the gases, cotton ("Molleton") which gave 0.07 per cent. ash and was free from H_2SO_4 and F. The material was cut into four parts—three-cornered pieces of about 250 sq. cm. The first were soaked in baryta-water and the last in lime-water. After drying, the bases were sufficiently fixed as carbonates. These cloths were hung up in the trees in the region of the smoke. After 5 to 7 months those soaked in baryta-water were examined for H_2SO_3[1] and those in lime-water for F., whereby 0.054 to 0.190 g. H_2SO_3 and 0.4 to 2.2 mg. F were obtained. Some years later he showed that these substances could be detected even in regions very remote from the source of injury. He took 0.130 to 0.180 g. H_2SO_3 after six months' exposure to represent the normal purity of the atmosphere of a German mountainous forest-region. In the barrens north of Hanover, which are free from all smoke, similar results were obtained. Near the city 0.79 g. was obtained. Similarly Ost carried out Wislicenus's experiments in the Saxon forest-districts by hanging in each district three prepared rags in such a position that the wind should have free access to them. Besides the chemical examination he made tests of the soot by comparisons of the color in clear daylight, made by three or four persons. In this way he distinguished six grades of smudging. Wislicenus thus summerized his conclusions in 1897:

1. Forest-air, even at great distances from sources of contamination, contains S acids.

2. $BaCO_3$ in 5.5 months was very nearly saturated by the air.

3. The higher degrees of saturation of $BaCO_3$ (which averages 94 per cent.) and of smudging were proportional to the extent of exposure.

4. Although SO_2 penetrates the thicker clumps of firs, it is not so much absorbed on account of lack of light.

5. Soot does not penetrate far into thick clumps of firs.

At first, experiments in rain-water and snow were thought important, but this is only in certain cases. Lately P. Sorauer (*Jahrsbericht für Agricultur Chemie*, Dritte Folge III. der ganzen Reihe 43er Jahrgang, p. 456 (1900)), proposed to employ plants for absorbing the injurious acids. A year's growth of *Phaseolus vulgaris* in the neighborhood of the suspected works was recommended for this.

All these methods are of doubtful value.

(Here follow tables of acute and of chronic injuries from smoke, in both of which wood-smoke is rated 0. Chronic injury is ascribed "almost exclusively" to SO_2.)

Special Part I. is devoted to the consideration of SO_2 and H_2SO_3

1. The means of condensing the acid-products from the gases of those establishments producing SO_2 may be estimated from the fact that the Freiberg smelting-works paid 55,000 marks for damage in 1864, and only 4,793 in 1870.

. . . In England it is enacted that the amount of S acids discharged into the air must not exceed 4 grains per cu. ft. (9.2 g. per cu. m.) calculated as SO_3. In fact they seldom reach 1.5 grains per cu. ft. The Prussian Minister of Commerce and Manufactures has forbidden that the content of SO_2 in chimney-gases when near inhabited dwellings shall exceed 0.02 per cent. in volume. Angus Smith has calculated that the H_2SO_4 in a million cu. m. of air amounts, in London, to 1,670 g.; in Manchester, to 2,518 g.; in smaller places where H_2SO_4 is manufactured, 2,668 g., and in places where coal is not employed, 474 grams.

Freytag estimates that in the year 1876 in Hanover and Linden 140,000,000 kg. of coal was burned and 2,100,000 kg. of SO_2 was discharged into the air. In the narrow

[1] Sulphurous acid, calculated from sulphuric acid.

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valley near Lethemathe on the Lenne, 846,000 kg. of S as SO_2 was discharged into the air in 1879. . . .

2. Besides SO_2, in the above cases, more or less SO_3 is formed, which in presence of steam and water is rapidly changed to H_2SO_4. The water-solution of SO_2 becomes rapidly H_2SO_4. Freytag obtained no SO_2 from rain-water in the vicinity of roasting-furnaces, though traces of this gas were present at the point of oxidation. W. Thörner found no SO_2 in the locomotive-smoke, but only H_2SO_4. . . .

3. SO_2 is never found in the soil. Freytag proved experimentally the rapid conversion of SO_2 to H_2SO_4 by contact with soil. Free H_2SO_4 is not found in soil. The acid reacts on the carbonates. Freytag found in certain places near Aachen in a liter of rain-water 9.0026 to 0.0069 g. HCl, 0.0031 to 0.0194 g. H_2SO_4, of which latter only 0.0038 to 0.0069 g. was free acid. Experiments by Freytag on snow in the neighborhood of the Halsbrücke and Muldner works show the same fact. The sulphates which accumulate in the ground, and especially the $CaSO_4$, are highly favorable to plant-growth, but being soluble, they are carried away by the ground-water.

Schroeder and Reuss proved that, roughly speaking, with the same amount of H_2SO_4 in the soils, the extent of the injury to the plants was measured by the amount of H_2SO_4 in the leaves and needles.

These examples prove that in spite of the strong and repeated influence of the sulphurous- or sulphuric-acid smoke-gases, whether direct or through atmospheric precipitation, no perceptible increase of the content of H_2SO_4 is observed. Hence the conclusion is to be drawn that, exclusive of the reactions of the soil, the S-acid smoke-gases produce no changes in the soil, and therefore there can be no injury to the soil by smoke-gases.

4. (*A*) *Action on the Subterranean Organs.*—From what has been said, it will appear that the action of SO_2 on the roots is excluded because of the rapid oxidation of SO_2 to H_2SO_4. As to the chance of this latter injuring the roots, it is recalled that experiments on snow and rain prove that the quantities of H_2SO_4 which can enter the soil are exceedingly small; of this none would be left uncombined with bases in the ordinary soil, and if one imagine a soil without bases to neutralize it, even then the percolation of the ground-water to lower levels would carry the acid quickly below the roots. In fact, free H_2SO_4 has hardly ever been observed in the soil.

Freytag watered summer wheat, oats and peas growing in beds, morning and evening with 20 liters of water containing, in one case, 4 g. (= 0.02 per cent. SO_2), in another, 5 g. (= 0.025 H_2SO_4), from May 1 to June 15. Then the acid was increased to 5 g. SO_2 and 7 g. H_2SO_4. From July 1 to July 14 a further increase to 8 g. SO_2 and 10 g. H_2SO_4 was made, without the least appearance of disturbance of the ordinary course of growth. On June 15 the oats and peas were gathered, and on July 31 the wheat was harvested. The analyses of the matured products showed no differences unfavorable to the treated plants. Later experiments with the stronger solutions proved that injury only began to be manifest when shortly after the watering a warm wind arose, which evaporated the water and concentrated the acid. As rain can never carry such quantities of acid into contact with the roots it must be conceded that the effect of the smoke-gases on the soil may be neglected.

Reuss fully proved these results by experiments on firs in the forest.

(*B*) *Influence on the Supraterraneous Organs.*—As a result of experiments of J. v. Schroeder and Schmitz-Dumont it follows that injury of plants by SO_2 or H_2SO_4 smoke-gases is always accompanied by an increase of the percentage of H_2SO_4 in the organ of the leaf.

From the foregoing experiments the following conclusions may be drawn:

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1. A direct action of SO_2 or H_2SO_4 on the roots of plants in normal farming and forestry conditions is unlikely.

2. An increase in the S-content of the soil through SO_2 and SO_3 gases is without influence on the growth of the plants, and therefore injury to the latter through the action of smoke-gases on the soil may be neglected.

3. An injurious effect on the plant can only occur when SO_2 and SO_3 of the smoke-gases come into actual contact with the leaf-organs of the plant. Concomitant with this injury is an increase in the H_2SO_4 content of the plants. But as the latter may be the result of an increased S-content of the soil only a consideration of the peculiar conditions of each place can properly interpret the phenomenon.

. . . Notwithstanding Stöckhardt convincingly proved that even the smallest amounts of SO_2, by frequent action could work injury to vegetation, Freytag threw doubt upon this (compare last chapter of *Wirkung der Feuchtigkeit und Trockenheit*). This doubt was finally removed by the experiments of J. v. Schroeder and W. Schmitz-Dumont. (*Tharander Förstliches Jahrbuch*, vol. xlvi., p. 1 (1896)). . . . In the determination of the H_2SO_4 in the plant-organs we have an essential means of proving the effect on vegetation of SO_2 and H_2SO_4. . . .

In the case of conifers, and probably other plants, of which the leaf-organs are gummy or waxy, the SO_2 or H_2SO_4 taken up through the rain will not be completely saturated, and thereby the determination of the smoke-injury will be impossible.

(*b*) Morphological Changes.—I. Experimental production of smoke-injuries. II. Exterior changes in the leaf-organs; III. Inner changes in the leaf-organs; IV. Alterations in the stem-organs: acute and chronic injuries.

(*c*) Physiological changes.

(*C*) (P. 130.) Influence of the action of SO_2 by various factors.

(*a*) Light.

(*b*) Influence of moisture and dryness.

(P. 136.) Moist, misty air with a content of 0.003 per cent. by weight, or 0.00135 per cent. by volume, of SO_2, is not injurious. Injury for vegetation has a boundary line between 0.003 and 0.004. . . .

From Freytag and Schroeder's experiments it appears that dryness protects from, and moisture exposes to, injury from smoke-gases. This result agrees with the practical experience that in mist and dew the injury is greater than when the weather is dry.

The influence of SO_2 is greatest when light, moisture and warmth are present.

(*c*) Influence of position.

(*D*) Influence of SO_2 on the cell.

5. *Résumé* of results of investigation (pp. 143–5, 14 conclusions).

1. Even with strong and repeated treatment of a soil by SO_2 and H_2SO_4 smoke-gases, either directly or through the atmosphere, no essential increase of the sulphur-content of the soil is effected. Disregarding the reactions of the constituent parts of the soil, no change of constitution of the soil takes place, and therefore injury to the soil by SO_2 or H_2SO_4 is out of the question.

2. A direct action of free SO_2 or H_2SO_4 of smoke-gases on the roots of plants is improbable in ordinary farm and forestry conditions. Should an increase of sulphates occur through the action of SO_2 or H_2SO_4 smoke-gases on the soil, it would have no effect on the growth of the plants, and therefore injury to plants through the action of acid smoke-gases on the soil may be excluded from consideration.

3. An injurious effect on plants can only occur when the acid-gases come into direct contact with the leaf-organs of plants. By injury of plants through SO_2 the content of H_2SO_4 in the plant is always increased, but as this occurs when the soil

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increases in its content of sulphates, the observation by itself cannot prove injury through acid-gases. The peculiar conditions of each place must be considered.

4. The sensitiveness of plants to SO_2 and H_2SO_4 varies. Even plants of the same kind are differently susceptible, according to their individual position.

5. After long continuous exposure, even so small a quantity as one-millionth of SO_2 has been found injurious to plants. According to Freytag H_2SO_4 is more, while J. v. Schroeder finds it less, injurious to plants than SO_2.

6. The amount of SO_2 collected from the same amount of leaf-surface of two different plants, under nearly the same circumstances, will of itself afford no measure of the injury done the whole organism of the plants; on the contrary, the specific peculiarities in the organization of the several plants must be taken into account and submitted to proof.

7. The cracks in the leaf-organs have nothing to do with the absorption of SO_2. The gas is not absorbed through these cracks, but by the entire leaf-surface; therefore the different quantities of SO_2 absorbed do not depend upon the number of the cracks but upon the peculiar organization of the individual plants.

8. The effect of the absorption of SO_2 is to disturb the circulation of water, which appears in an increased extrusion of water, and leads to the drying of the leaves.

9. The absorption of SO_2, and consequent disturbance of the water circulation, is for the same quantity of SO_2 greater in a given time with light, higher temperature, and dry air than in darkness, lower temperature, and moist air. The SO_2 and acid gases are in general more injurious by day than by night.

10. Morphologically, the effect of SO_2 is shown by the formation of spots on the leaves, the death of leaves and twigs, the retardation of the rings of growth, and at last the destruction of the plant.

11. In the interior of the cell plasmolysis is induced, the grains of chlorophyll are destroyed, and finally form with the plasma and the other constituent materials a brown amorphous mass. At the same time in most cases, especially if the injury has been gradual, tannin separates out as brown or black rolls in the cells.

12. The manner of action of the SO_2 is to be figured as a disturbance of the life of the plasma in the cell. It probably acts as H_2SO_4, produced in the oxidation of SO_2 by the oxygen of the assimilating chlorophyll-grains in presence of water from the cell-sap.

13. By continuous action of water or rain the SO_3 or H_2SO_4 of the dead leaf-organs, which has been taken out of the air, may be again eliminated. In the conifers, and probably other plants, of which the organs are gummy or waxy, the SO_2 or H_2SO_4 taken from the smoke-gases is not further neutralized in the mass, so that the recognition of smoke-injury is impossible.

14. No absolutely sure botanical means of recognizing the injuries of SO_2 exists, but it is only possible through the complex of outer and inner injuries to conclude their presence. The surest proof is the chemical analysis for H_2SO_4. . . .

6. Instances from actual practice.

(P. 177.) . . . The Freiberg smelting-works were scientifically examined in the middle of the Nineteenth Century, on account of complaints of farmers of the death of plants and cattle. So long as the metallic components of the smoke were held responsible for the damage not much progress was made.

A. Stöckhardt was the first to recognize that SO_2 was the injurious agent. This caused the erection of the H_2SO_4 works, now in operation for 40 years, which utilizes the greater part of SO_2. Yearly the injury diminished, until in the eighties a narrow strip next to the works alone suffered damage and the State purchased it.

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The Tharand wood lies in such a position that westerly winds blow the smoke from the Freiberg works over it.

Inclosing by a line the firs of this wood in which 0.25 per cent. H_2SO_4 has been detected, the area forms a flat ellipse, to the south of which lies the Muldner, and in the middle the Halsbrucker works are situated. To the east toward the Tharand forest the ellipse continues in two gigantic rounded terminations.

The high content of H_2SO_4 near the railways is worthy of remark.

Stöckhardt first showed that the coal-smoke from locomotives contains SO_2, and therefore may produce injury to vegetation. . . .

(P. 304.) Fog. (See report of F. Oliver, *Journal of the London Horticultural Society*, vol. xiii., p. 139 (1891), and vol. xvi., p. 1 (1893)).

Chapter XI. Lighting-gas Very Injurious to Roots. — Experiments by Wehmer in Hanover, and L. Kny (*Botanische Zeitung*, vol. xxix., pp. 852, 867 (1871)), in the botanical garden of Berlin: The supraterranean parts of the plant are seldom damaged. The dead roots are bluish in the interior. As the intensity of the color diminishes towards the periphery Kny concludes the gas is introduced dissolved in water through the root-tips. The phenomenon is not always observable and the conclusion needs further proof.

Chapter XII. Comparison of the Injurious Effects of Acid-Gases.—Tables of minimum and maximum emission of HCl, SO_2, and H_2SO_4 gases of 40 alkali-works by Angus Smith. Also tables of lead-works, smelters, etc.

Turner and Christison think HCl more injurious than SO_2.

Richardson concludes from his experiments that Cl is the most intense in its action, SO_2 next, and HCl least.

Angus Smith concludes from experiments on water-plants with very dilute, but equally strong, solutions of SO_2, H_2SO_4, and HCl, that H_2SO_4 is the most injurious, HCl next, H_2SO_3 least.

Freytag reaches the same conclusion, and adds that H_2SO_3 is injurious only because it is oxidixed in the moist chlorophyll green leaves to H_2SO_4, which on concentration corrodes. But the results of the experiments of v. Schroeder are opposed to all these views

It cannot be doubted, judging by the action of equal quantities of the beforementioned acids, that H_2SO_3 is the most injurious, and H_2SO_4 and HCl less so. v. Schroeder and Reuss justly point out that the relations are very different when all these gases are simultaneously emitted from a chimney. In such a case HCl and H_2SO_4 condense more quickly, while H_2SO_3 is carried farther. The first does act most strongly on the vegetation in the immediate vicinity, while the H_2SO_3, from its less solubility and consequent slower condensation, has a wider distribution.

H. Ost believes that F gas is much the most injurious of all.

Chapter XIII. Volatile Dust (*Flugstaub*). . . .

4. Effect upon cattle. The dust contains no nourishment and its sharp and pointed particles may cause injury to the intestines of the cattle. Haubner (*Archiv für wissenschaft und praktische Thierheilkunde*, pp. 97, 241 (1878)), made many thorough experiments in connection with cows near the Freiberg works.

The cattle feeding on grass reached by the smelting-works gases suffered from a disease. The milch-cows using this fodder gave little milk, and that poor in fat, and after calving the separation of the milk lasted a shorter time. Haubner divided the sickness as follows:

(*a*) So called acid-sickness, a kind of disease of the bones or of the marrow-fluids, caused by the effect of the acids on the fodder-plants.

(*b*) Lung tuberculosis, with its premonitory symptoms, tracheal and bronchial catarrh, cheesy pneumonia.

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(*c*) Inflammatory conditions and spasms (*Quetschungen*) of the stomach, and the perforation of the larger intestine (*Labmagen*). The last two are caused by the emitted furnace-dust.

The symptoms of the acid-sickness are: Frequent passages (with acid reaction), pallor and hard skin. The visible mucous and conjunctiva become noticeably pale, the skin becomes dry, hard, and immobile, especially in the region of the barrel (*Rippengewölbe*), is dusty and uncleanly; the hair is lusterless, rough and tangled; besides this comes later a diminution of the appetite, and of the milk, and gradual attenuation. The urine is remarkably pale, clear, like water, without deposit, and with acid reaction. The animals stand with lowered head and neck, cannot sufficiently raise them. The back is bent and the belly lowered. The hind-quarters take a straight, stiff position in all joints, which in the hock (*Fesselgelenk*) is first manifested by a stiff position of the pastern. Then follow the hip (*Sprung*) and hind knee-joints, so that the angles continually diminish. Later symptoms appear which indicate disease of the bones, as occasional pains in the joints, indicated by stiffness and difficulty in moving, etc.

These appearances are seen more clearly in young than in old cattle. Freytag thinks these conclusions go too far and Haselhoff agrees with him.

General remarks on smoke-expertism are summarized as follows:

Chapter I. R. Hartig thinks the measure of injury to vegetation by the determination of H_2SO_4 has merely an historic interest.

Borggreve (*Waldschäden im oberschlesischen Industriebezirk*, Frankfort (1895)), says "by these eternal commonplace H_2SO_4 determinations" no more is proven than we have long known. J. v. Schroeder has justly and sharply retorted to this sneer. He says: "If in any future judicial process I should have Borggreve's objections opposed to any conclusions I should draw from sulphuric-acid determinations, I should simply say that Borggreve says of himself, in several parts of his book, that he is not enough of a chemist to judge of chemical methods and understands little or nothing of chemistry."

Chapter II. Botanical Examination.

Chapter III. Examination of the locality, and Selection of specimens.

For forest- and fruit-plants, the middle of July is the best time (in Germany). For field-plants, the middle of June.

It is useful to take specimens from the greatest number of directions.

Wislicenus suggests the observation of the top of the chimney to ascertain which is the prevailing direction of the wind by the greater deposit of soot on the side of the chimney most protected from the wind.

Chapter IV. Chemical Examination.

In preparing the specimen for analysis it must be carefully cleaned of sand-particles and brushed with a fine brush or rinsed with water. Plants holding much water must be dried at 50° or 60°. After a thorough mixing, the whole sample should be comminuted as finely as possible in an Excelsior mill, or if that is not enough, in a mortar.

Chapter V. Botanical Examination.

Chapter VI. Estimation and Prevention of Injury.

Weslicenus's formula for injury is $S = \sqrt[2]{\frac{c.kw.}{d^3}} \times \frac{C}{X}$

S = proportion of injury; c = acid-content of the smoke-gases in volume.

k = yearly consumption of coal; w = particular percentage of direction of the wind; d = minimum distance; C = constant of the injurious kind of gas.

X = factor for the reduction of the distance by the influence of acid-fogs, etc.

Haselhoff gives the formula without knowing anything of its reliability.

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By the prevention of smoke an industrial works can only be the gainer, and generally by the prevention of the escape of acid gases.

It is not practicable to establish a permissible percentage of injurious products which may be emitted for all parts of a country. In Germany each separate case is judged by itself. As in Germany the prevailing winds are west and south, concessions for industrial establishments to the west and south of forest or cultivated lands should only be granted at considerable distances. In hilly regions the direction of the wind is to be more carefully considered than in flat.

45. SMOKE AND ITS ABATEMENT. Prof. C. H. Benjamin, *Transactions of the American Society of Mechanical Engineers*, vol. xxvi., p. 713 (1904–05).

Bituminous coal must be used, and no legislation can prevent it. Other fuels are in limited supply. Abatement is possible; prevention of its injuries is not. (*a*) Smoke is a nuisance, (*b*) can be easily abated, and (*c*) this abatement may be made the source of profit.

Black smoke is due to hydrocarbons in the fuel not having been provided with sufficient oxygen to consume them. The hydrogen is burned, and the carbon is carried off as soot. When supplied with enough oxygen, it burns with a yellow flame; when with too little, a reddish flame; soot is then formed. The conditions of perfect combustion are sufficient air, sustained high temperature, and thorough firing of the gas. Trees and shrubs are killed by the sulphurous content of smoke. With hand-firing great irregularities are experienced. Steam-jets are employed: they should be semi-automatic. The best solution of the smoke problem is mechanical handling of coal.

Mechanical stokers may be divided into: inclined; shaking grates; traveling, or chain grates; and underfeed stokers.

Inclined.—Wilkinson, Brightman, and Roney stokers, single incline; Murphy and Detroit stokers, double incline. Disadvantages: Both forms need frequent cleaning, and with clinker-making coal too much slicing.

Traveling, or Chain-Grate.—Babcock, Wilcox and Green.

Benjamin thinks this is the best form of grate used.

Underfeed Stokers.—The American and the Jones. Economical and practically smokeless.

A smoky chimney is an indication of waste, but a smokeless chimney is not necessarily an indication of economy.

Letters from persons using mechanical stokers favor this system.

(Table of efficiency of combustion.)

A recent improvement that promises well is a combination of steam-jets and oil-vapor at the bridge-wall. Baffle-walls have also assisted in maintaining high temperature and mixing the gases.

(Rules for firing in a locomotive by a railroad.)

46. THE ADVANTAGE OF A SCIENTIFIC BASIS FOR DETERMINING THE VALUE OF FUELS. Henry J. Williams (chemical engineer), *Journal of the New England Waterworks Association*, vol. xix., No. 1. (Paper read Dec. 12, 1904. Followed by a discussion.)

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The Wilfley Table, I.

BY ROBERT H. RICHARDS, BOSTON, MASS.

(Toronto Meeting, July, 1907.)

THIS truly remarkable machine was built on a preliminary scale in May, 1895. The first full-sized table was built by Mr. A. R. Wilfley, and was used in his own mill in Kokomo Colo., in May, 1896. The first table sold for installation was placed in the Puzzle mill, Breckinridge, Summit county, Colo., in August, 1896.

The mill enthusiasts at first hailed it as the cure for all the ills that flesh is heir to in the milling line. A little later it was found to make losses which were serious, and on this account the table succeeded only to a limited extent in displacing the vanners of the gold-mills. Still later, mill-men in a number of districts throughout the country made special studies of the faults of the machine, and devised a number of ways of grouping supplementary machines to overcome as far as possible the losses, and at the same time retain the benefit of the extraordinarily large capacity accompanied by the production of clean concentrates for which the machine has become so justly famous. I hope to make an exhibit of some of these methods in an appendix to my book on ore-dressing which is now in preparation. These experimenters have not written up the subject, and if they possess all the facts they have not given them out for the benefit of the mining profession at large.

The object of this paper is to obtain the facts and to present them so clearly that their bearing can be seen by all. To this end two complete series of tests have been planned. One (the present paper), to study concentration of galena in presence of quartz; the other (to follow shortly), to study concentration of chalcopyrite in presence of quartz.

Some authorities claim that the table does its best work when treating natural products; by this phrase I mean products which have been crushed to pass through a limiting

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sieve, but have had no other preparation whatever; in consequence they have all sizes of grains of both the heavy valuable mineral and the light waste gangue, ranging from the largest grains that can pass through the sieve down to the finest dust.

Others claim that the ore fed to a Wilfley table should be closely sized before it is fed. That is to say, it should be divided by a series of sieves ranging from coarse to fine into a series of products with sizes of grain ranging from coarse grains to fine grains; and that each of these products, in which the grains of the heavy mineral are of approximately the same diameter as the grains of the light mineral, should be fed to the Wilfley table.

Still a third group of authorities claims that the ore before being fed to a Wilfley table should be classified by a hydraulic classifier, which divides the crushed ore into a series of products ranging, like the sized products, from coarse grains to fine grains, by carrying it in a water-current over a series of apertures or vertical pipes, called sorting-columns, up through which water-currents are passing. These currents are graded from faster to slower, and therefore allow only the heaviest grains to settle down through the first sorting-column and out through the spigot, while lighter smaller grains settle in the second, and still lighter in the third, and so on, diminishing until the last sorting-column and spigot give very small grains, and the overflow has the finest grains of all. The classified products differ greatly from the sized products in that the grains of heavy mineral are much smaller in diameter than the light grains with which they settle, and therefore behave in a somewhat different way upon the Wilfley table from the sized products. It should be said that the first spigot-product of a classifier differs from the others in having coarse grains of heavy mineral present also.

The usual division of products upon a Wilfley table is easily and naturally made, as shown in Fig. 1, *A* being concentrates; *B*, middlings; *C*, tailings, and *D*, slimes. Of these, when natural products are fed, the concentrates, *A*, are nearly clean heavy mineral, a slight contamination of small grains of quartz being present. The middlings, *B*, carry some large grains and also some small grains of heavy mineral. The tail-

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ings, *C*, carry some very small grains of heavy mineral, and the slimes, *D*, carry very minute grains of heavy mineral.

I believe that the small grains of heavy mineral in middlings, *B*, and tailings, *C*, are of less diameter than the smallest in the concentrates, *A*, and of greater diameter than the majority in the slimes, *D*, and that they belong in middlings and tailings from the law of their existence. The re-running of such middlings upon the same table is therefore not a wise proceeding, and only admissible as an expedient in small establishments when the quantity of middlings is not sufficient to warrant other provision. So much for the speculation before the investigation was made.

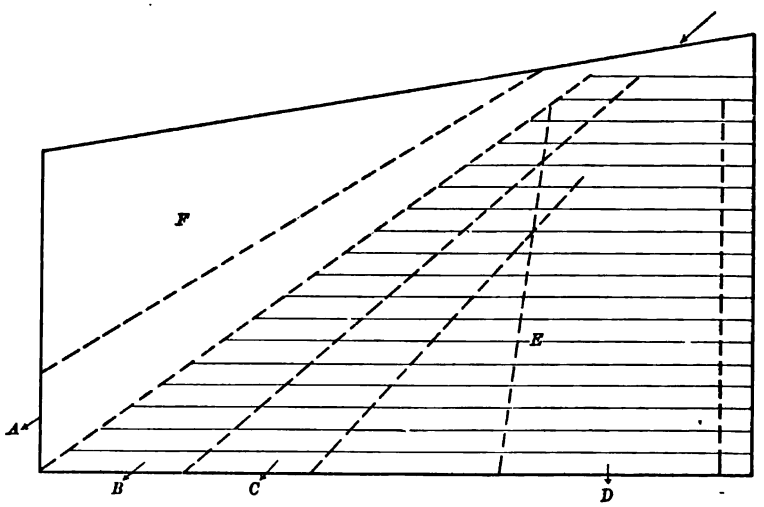

FIG. 1.—USUAL DIVISION OF PRODUCTS ON A WILFLEY TABLE.

The materials for this test were pure white massive quartz for the light mineral, and crystalline galena, nearly free from blende and other impurities, from Joplin, Mo., for the heavy mineral. The quantities of these impurities were so small as to have little effect on the results. Both minerals were broken down to 2-mm. size, and mixed so as to have approximately 10 per cent. of galena and 90 per cent. of quartz.

The Wilfley table used for the tests had a net working-surface of 2 ft. by 4 ft. This is the table that has been found very satisfactory for students' work at the Massachusetts Institute of Technology. An error is present to a slight extent in the full-sized table, 16 ft. long by 7 ft. wide, and to a serious ex-

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tent in the small Wilfley testing-table, 7 ft. long, found in some of the schools. It is due to tacking tapered riffle-cleats on to a linoleum plane surface, thereby making two planes: first, the roughing-plane over the riffle-cleats, *E*, Fig. 1; second, the cleaning-plane or part where there are no riffles, *F*, Fig. 1. These two planes make an angle with each other, or a slight trough, which heaps up the sands deeper than is wise along the line of the tips of the riffle-cleats. This error is completely overcome on the little table here used by cutting the riffles down into the wooden surface of the table. The roughing- and cleaning-planes are therefore one and the same plane. By observing this precaution I believe that this little table is able to do as good work as the full-sized table.

Comparing the little table with the full-sized table as to areas and capacity, assuming that their capacities are proportional to their areas, we have Table I.

TABLE I.—*Area and Capacity of Small and Large Wilfley Tables.*

	Little Table.		Large Table.	
Area, . . .	8	sq. ft.	112	sq. ft.
Feed, . . .	1	kg. per min.	22	tons per 24 hr.
Feed, . . .	0.75	kg. per min.	16.6	tons per 24 hr.
Feed, . . .	0.5	kg. per min.	11	tons per 24 hr.

These figures represent the usual range used in practice.

Seventeen runs in all were made; Nos. 1 to 5, inclusive, were made upon natural products, the several feed-products being 2 mm. to 0; 1 mm. to 0; 0.5 mm. to 0; 0.25 mm. to 0; and 2 mm. to 0. Run No. 5, although fed with the same size as No. 1, was fed at a different rate. In making these runs no effort was made to re-run the middlings; first, because the concentrates and tailings would both have been contaminated and would not have shown as well; second, because the middlings themselves would have undergone a change in composition. In consequence of this ruling, the quantities of middlings appear abnormally large.

In these runs the dividing-line between concentrates and middlings was chosen so as to make concentrates nearly clean to the eye. The dividing-line between middlings and tailings was chosen so as to keep all the large grains of heavy mineral in the middlings. The four products—concentrates, middlings,

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tailings and slimes—were sized on a series of sieves, and the quartz in them determined by dissolving out the galena in hydrochloric acid. The galena was determined by difference.

Runs Nos. 6 to 11, inclusive, were all upon sized products, and the results obtained are given in Table II.

TABLE II.—*Sizes and Weights of Materials Fed to Table. Tests Nos. 6 to 11.*

Run Number.	Sieves Diameter. Through. mm.	On. mm.	Actual Weights Fed. kg.
6	2	1.4	12.15
7	1.4	1.0	6.74
8	1	0.75	4.93
9	0.75	0.50	2.85
10	0.50	0.36	1.70
11	0.36	0.28	1.55
	0.28	0.00	3.08
			33.00

The total quantity weighed 33 kg., of which 30 kg. was quartz and 3 kg. was galena. The guiding was done simply to make clean concentrates and tailings. The middlings were in every case re-run until they could not be further reduced without contaminating the concentrates or the tailings. Where a sized ore is free from included grains and from any middleweight mineral, the feeding-back of the middlings on the same table is logically good practice, because the middlings product is simply a mixture of concentrates and tailings; therefore, they could be fed back on the same table and disappear entirely without harm to concentrates or tailings.

Runs Nos. 12 to 17, inclusive, were made upon sorted or classified products. The classifier, Fig. 2, had 12 closed spigots or blind spigots; that is to say, spigots which discharged sand into 2-gallon bottles as fast as it came, but discharged no water. The sorting-columns were of 0.5-in. pipe, squared at the top and 3 in. long. Expressed in mm. per second, the rising-currents in the successive sorting-columns were: 105, 85, 69, 55, 45, 36, 29, 23, 19, 15, 12, 10, respectively. The 13th spigot had no rising-current, and it was simply a safety spigot to prevent any accumulation of sand that was too light to go down in the 12th and too heavy to go over into the overflow. This apparatus gives a set of products beautifully classified.

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To define the classified products more completely, a small aliquot part of each was sized, photographed and analyzed. The photograph, Fig. 3, shows to the eye the distribution of sizes in each spigot. Table III. shows the distribution of quartz and galena in the different sizes of each spigot. Expressed in mm., the sieve-sizes used were: 2.83, 2.49, 2.06, 1.63, 1.44, 1.27, 1.10, 0.97, 0.84, 0.68, 0.57, 0.45, 0.36, 0.28, 0.24, 0.20, 0.15, 0.12, 0.10, 0.08. The middlings picked out and weighed for the first four spigots consisted of blende-galena included

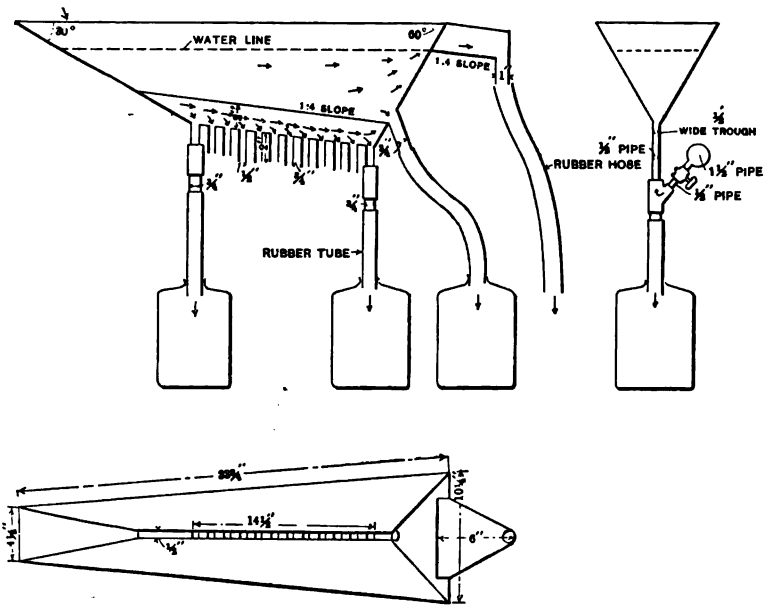

FIG. 2.—CLASSIFIER USED FOR RUNS NOS. 12 TO 17.

grains, with some free blende-grains. Their use here is simply to show how nearly pure the galena was. The settling-ratios are of special interest; for example, at the foot of spigot No. 4 we have the settling-ratio, 3.26, which signifies that in spigot No. 4 the average diameter of the quartz-grains is 3.26 times the average diameter of the galena-grains. The method of computing these ratios is given in my paper, Close Sizing Before Jigging.[1] It was thought wiser to combine the spigots

[1] *Trans.*, xxiv., 449, 450 (1894).

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TABLE III.—*Distribution of Quartz and Galena in Runs Nos.* 12 *to* 17.

Run No.		12			13			14					
No. of Spigot		1			2			3			4		
Actual Weight in kg.		15.110			15.775			11.175			5.170		
Rising Current, mm. per sec.		105.			85.			69.			55.		
Through mm.	On mm.	Quartz.	Galena.	Middlings.	Quartz.	Galena.	Middlings.	Quartz.	Galena.	Middlings.	Quartz.	Galena.	Middlings.
2.83	2.49	0.052	0.004	0.007	.....	.....	.....	.....	.....	.....	.....	.....	.....
2.49	2.06	2.408	0.709	0.021	1.517	.....	.....	0.213	.....	.....	.....	.....	.....
2.06	1.63	17.170	7.216	0.085	20.299	.....	0.006	3.617	.....	0.003	0.112	.....	.....
1.63	1.44	9.885	7.650	0.112	20.805	0.004	0.009	6.510	0.002	0.003	0.394	.....	.....
1.44	1.27	9.381	8.476	0.118	19.856	0.003	0.013	12.635	0.003	0.004	1.210	.....	.....
1.27	1.10	2.480	3.391	0.010	15.746	0.004	0.016	14.249	0.002	0.002	3.475	.....	0.001
1.10	0.97	1.945	3.742	0.022	5.161	0.006	0.011	14.339	0.003	0.001	2.573	.....	0.001
0.97	0.84	2.041	5.435	0.025	5.925	0.037	0.023	15.208	0.027	.....	9.530	0.006	.....
0.84	0.68	0.861	3.914	0.020	2.734	0.063	0.017	10.646	0.018	.....	12.793	0.008	.....
0.68	0.57	0.535	3.479	0.009	1.527	0.147	0.013	7.499	0.030	.....	20.128	0.037	.....
0.57	0.45	0.356	4.143	.....	1.558	0.865	.....	8.795	0.167	.....	26.312	0.056	.....
0.45	0.36	0.100	1.634	.....	0.217	0.678	.....	1.437	0.203	.....	9.741	0.070	.....
0.36	0.28	0.068	1.407	.....	0.107	1.215	.....	0.942	0.678	.....	6.502	0.275	.....
0.28	0.24	0.022	0.457	.....	0.028	0.517	.....	0.190	0.455	.....	1.638	0.489	.....
0.24	0.20	0.016	0.258	.....	0.015	0.414	.....	0.124	0.825	.....	0.667	0.892	.....
0.20	0.15	0.010	0.145	.....	0.009	0.233	.....	0.045	0.594	.....	0.223	1.107	.....
0.15	0.12	0.007	0.089	.....	0.005	0.107	.....	0.013	0.295	.....	0.068	0.715	.....
0.12	0.10	0.001	0.018	.....	0.003	0.025	.....	0.007	0.093	.....	0.037	0.458	.....
0.10	0.08	0.007	0.037	.....	0.005	0.029	.....	0.010	0.102	.....	0.025	0.301	.....
0.08	0.00	0.007	0.057	.....	0.009	0.023	.....	0.090	0.050	.....	0.012	0.132	.....
Total		47.352	52.261	0.429	95.526	4.370	0.108	96.569	3.587	0.013	95.440	4.546	0.002
Settling-ratio		1.58			4.11			3.93			3.26		

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TABLE III.—*Continued.*

Run No.		15				16						17					
No. of Spigot		5		6		7		8		9		10		11		12		Overflow.	
Actual Weight in kg.		2.280		1.950		1.182		0.389		0.962		0.765		0.651		0.705		1.860	
Rising Current, mm. per sec.		45.		36.		29.		23.		19.		15.		12.		10.			
Thro'h mm.	On mm.	Quartz.	Galena.	Quartz.	Galena.	Quartz.	Galena.	Quartz.	Galena.	Quartz.	Galena.	Quartz.	Galena.	Quartz.	Galena.	Quartz.	Galena.	Quartz	Galena.
2.83	2.49	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....
2.49	2.06	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....
2.06	1.63	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....
1.63	1.44	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....
1.44	1.27	0.031	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....
1.27	1.10	0.132	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....
1.10	0.97	0.184	.....	0.015	.....	0.008	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....
0.97	0.84	1.467	.....	0.082	.....	0.017	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....
0.84	0.68	5.161	.....	0.949	.....	0.042	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....
0.68	0.57	5.200	0.009	1.668	.....	0.220	.....	.....	.....	0.010	.....	.....	.....	.....	.....	.....	.....	.....	.....
0.57	0.45	29.990	0.057	16.732	.....	3.133	.....	0.150	0.101	0.073	.....	0.026	.....	.....	.....	.....	.....	.....	.....
0.45	0.36	25.038	0.026	18.590	0.380	9.086	0.084	5.156	0.125	1.850	0.135	0.040	.....	.....	.....	.....	.....	.....	.....
0.36	0.28	19.164	0.119	34.011	0.272	36.110	0.338	20.250	0.125	13.185	0.260	6.799	0.171	1.251	0.015	0.114	0.014	0.169	0.238
0.28	0.24	4.779	0.132	9.752	0.082	15.539	0.160	23.252	0.225	14.806	0.291	3.980	0.026	3.961	0.104	0.528	0.057	0.185	0.195
0.24	0.20	2.280	0.435	8.884	0.257	17.499	0.253	26.435	0.350	29.790	0.364	23.061	0.132	12.375	0.134	5.083	0.129	0.723	0.486
0.20	0.15	0.725	1.019	2.171	0.406	7.127	0.296	12.740	0.275	17.320	0.281	29.820	0.211	25.773	0.223	11.593	1.157	2.243	0.438
0.15	0.12	0.242	1.612	1.011	1.273	3.344	0.828	3.755	0.325	11.813	0.416	18.685	0.303	26.653	0.298	21.415	1.999	10.116	0.950
0.12	0 10	0.119	0.663	0.205	1.232	0.650	0.769	0.901	0.526	2.275	0.499	5.455	0.329	11.615	0.268	11.464	1.171	8.575	0.522
0.10	0.08	0.018	0.474	0.092	1.140	0.507	2.111	0.951	2.203	1.299	2.005	3.321	1.186	7.787	0.819	20.500	2.413	28.520	1.805
0.08	0.00	0.048	0.879	0.046	0.739	0.228	1.647	0.225	1.902	0.416	2.910	1.700	4.744	3.529	5.182	17.475	4.982	35.460	9.372
Total		94.578	5.425	94.208	5.781	93.510	6.486	93.815	6.157	92.837	6.261	92.887	7.102	92.944	7.043	88.172	11.922	85.991	14.006
Settling-ratio.		3.19		2.62		2.49		2.21		2.21		2.74		2.46		1.28		1.10	

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somewhat instead of making 12 separate runs; accordingly, the 6 runs were fed with products as follows: 1st spigot; 2d spigot; 3d and 4th together; 5th and 6th together; 7th, 8th and 9th together; 10th, 11th and 12th together.

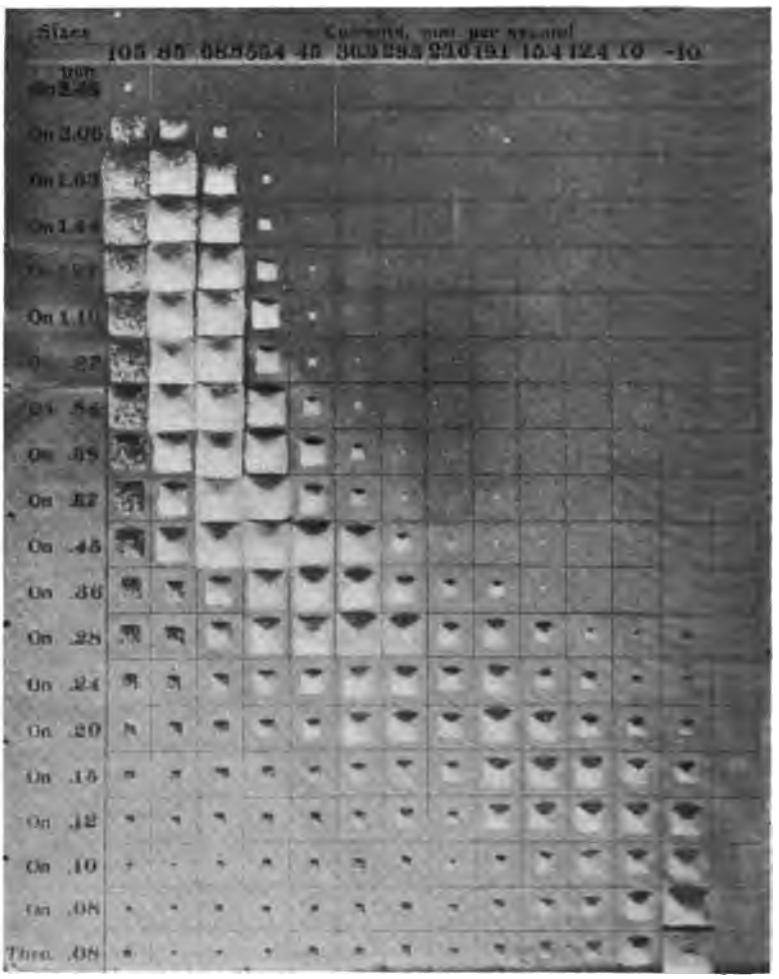

FIG. 3.—DISTRIBUTION OF SIZES BY THE CLASSIFIER.

The first 5 runs on natural products computed on the basis of a 100-ton lot, and to the rate of feeding of a full-sized table, gave products shown in Table IV.

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TABLE IV.—*Results of Runs Nos.* 1 *to* 5.

	Run No. 1.	Run No. 2.	Run No. 3.	Run No. 4.	Run No. 5.
Size of Feed.	mm. 2 to 0.	mm. 1 to 0.	mm. 0.5 to 0.	mm. 0.25 to 0.	mm. 2 to 0.
	Tons.	Tons.	Tons.	Tons.	Tons.
Rate of feed per 24 hours, .	22.	22.	11.	11.	11.
Concentrates, . . .	4.114	2.973	4.667	5.343	2.808
Middlings, ...	21.486	20.750	25.981	21.028	45.538
Tailings, ...	72.856	72.903	62.813	60.487	50.688
Slimes, ...	1.544	3.374	6.539	13.142	0.966
Total, ...	100.000	100.000	100.000	100.000	100.000

The next 6 runs on sized products, computed on the basis of a 100-ton lot, and to the rate of feeding of a full-sized table, gave products shown in Table V.

TABLE V.—*Results of Runs Nos.* 6 *to* 11.

	Run No. 6.	Run No. 7.	Run No. 8.	Run No. 9.	Run No. 10.	Run No. 11.
Size of Feed, . .	mm. 2.00 to 1.4	mm. 1.4 to 1.0	mm. 1.0 to 0.75	mm. 0.75 to 0.50	mm. 0.50 to 0.36	mm. 0.36 to 0.28
	Tons.	Tons.	Tons.	Tons.	Tons.	Tons.
Rate of feed per 24 hours, . . .	22.	16.6	11.	11.	11.	11.
Concentrates, . .	6.537	9.728	12.064	13.168	12.171	15.137
Middlings, . .	1.647	1.901	1.694	1.602	5.310	3.170
Tailings, . .	91.816	88.371	86.242	85.230	82.519	81.693
Slimes, . . .	0.000	0.000	0.000	0.000	0.000	0.000
Total, . .	100.000	100.000	100.000	100.000	100.000	100.000

The last 6 runs, computed on the basis of 100-ton lots, and to the rate of feeding of a full-sized table, gave products shown in Table VI.

TABLE VI.—*Results of Runs Nos.* 12 *to* 17 *Computed to Full-Sized Table.*

	Run No. 12.	Run No. 13.	Run No. 14.	Run No. 15.	Run No. 16.	Run No. 17.
Classifier spigots, numbers that were fed, .	1.	2.	3, 4.	5, 6.	7, 8, 9.	10, 11, 12.
Classifier currents, mm. per second through which grains settled,	105.	85.	55.4	36.3	19.1	10.
	Tons.	Tons.	Tons.	Tons.	Tons.	Tons.
Rate of feed to Wilfley, 24 hours, .	11.	11.	16.6	11.	11.	11.
Concentrates, . .	49.649	3.950	3.398	4.940	5.7165	4.980
Middlings, . . .	5.534	1.307	0.663	0.828	1.0435	4.262
Tailings, . . .	44.817	94.743	95.939	94.232	93.240	90.758
Slimes, . . .	0.000	0.000	0.000	0.000	0.000	0.000
Total, . . .	100.000	100.000	100.000	100.000	100.000	100.000

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Comparing the 17 runs as to quantity of the products without looking at the quality, it will be noticed at once that the concentrates and tailings in runs Nos. 6 to 11 and Nos. 12 to 17 are very much larger in quantity than these products in runs Nos. 1 to 5; while the middlings are very much smaller in quantity.

A comprehensive table of all 17 runs is given in Table VII., which shows the proportions of concentrates, middlings, tailings and slimes in each, and also the percentage of galena and quartz in the various products.

Tables VIII., IX., X. and XI. give the weights in tons of the different sizes, and also the proportion of quartz and galena in each product by sizes.

In comparing the analyses of the concentrates of these five runs (Table VIII.), we see a very remarkable similarity in the behavior of the quartz and galena through all five runs. The coarser sizes and the finer sizes are almost clean galena, being nearly free from quartz. At a point somewhere a little below the middle, the quartz rises to a maximum, which in the first run reaches 18.99 per cent. of quartz, in the third run reaches 7.38 per cent. of quartz, and in the fifth run reaches 13.80 per cent. of quartz.

Comparing the different analyses of the middlings (Table IX.), we find the galena among the very largest and smallest grains gives a very high percentage, and down to a little below the middle the galena runs down to a very low percentage, while the quartz behaves just in the opposite way. This is analogous to the composition of the concentrates.

Comparing the tailings of the five runs (Table X.), we note that the galena appears only to a very slight degree in any of the tailings until we get down to the smaller sizes, and there we have figures that rise to an almost alarming size, the first run giving 17.5 per cent. of galena in the finest size; the 2d, 17.8 per cent.; the 5th run giving 12.17 per cent. in the finest size.

The slimes (Table XI.), which have a serious quantity of material only in the finest size, have also a serious percentage of lead in that finest size. The other percentages of lead are generally much smaller. There are three exceptions, in the 2d, 3d and 4th runs, where the percentages run high in the larger

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TABLE VII.—*Summary of Results of Runs Nos.* 1 *to* 17.

No. of Run.	Size.	Rate of Feeding.		Composition of the Feed.		Percentage of the Products.				Composition of Products.								Per Cent. of Total Galena.				Total Quantity Actually Fed.
		Kg. Per Min.	Tons 24 hrs. la'ge Table.	Quartz.	Galena.	Concentrates.	Middlings.	Tailings.	Slimes.	Concentrates.		Middlings.		Tailings.		Slimes.		Concentrates.	Middlings.	Tailings.	Slimes.	
										Quartz.	Galena.	Quartz.	Galena.	Quartz.	Galena.	Quartz.	Galena.					
	mm.			Per Ct.	Per Ct.	Per Ct.	Per Ct.	Per Ct.	Per Ct.	Per Ct.	Per Ct.	Per Ct.	Per Ct.	Per Ct.	Per Ct.	Per Ct.	Per Ct.	Per Ct.	Per Ct.	Per Ct.	Per Ct.	Kg.
1	2 to 0	1	22	90.91	9.09	4.11	21.48	72.87	1.54	9.92	90.08	76.42	23.58	99.49	0.51	81.10	18.90	39.20	53.34	3.82	3.64	11
2	1 to 0	1	22	90.91	9.09	2.97	20.75	72.91	3.37	8.50	91.50	73.37	26.63	99.39	0.61	90.29	9.71	34.58	57.34	4.72	3.36	11
3	0.5 to 0	0.5	11	90.91	9.09	4.67	25.98	62.81	6.54	2.40	97.60	84.10	15.90	99.10	0.90	87.82	12.18	44.86	41.98	5.44	7.72	11
4	0.25 to 0	0.5	11	90.91	9.09	5.34	21.03	60.49	13.14	2.27	97.73	87.16	12.84	98.39	1.61	90.04	9.96	51.48	27.59	8.20	12.73	11
5	2 to 0	0.5	11	90.91	9.09	2.80	45.67	50.30	1.23	4.66	95.34	86.29	13.71	99.71	0.29	85.38	14.62	29.06	67.79	1.62	1.53	9.63
6	2 to 1.4	1	22	92.60	7.40	6.54	1.65	91.81	.....	0.77	99.23	40.33	59.67	99.96	0.04	.....	.....	87.43	12.05	0.52	.....	12.08
7	1.4 to 1	0.75	16.6	89.08	10.92	9.73	1.90	88.37	.....	0.77	99.23	76.90	23.10	100.00	0.00	.....	.....	96.43	3.57	0.00	.....	6.74
8	1 to 0.75	0.5	11	87.32	12.68	12.06	1.69	86.25	.....	0.99	99.01	83.00	17.00	99.91	0.09	.. ..	.....	97.62	1.80	0.58	.....	4.93
9	0.75 to 0.5	0.5	11	87.15	12.85	13.15	1.60	85.25	.. ..	2.50	97.50	77.40	22.60	99.65	0.35	.....	.....	96.58	2.29	1.13	.....	2.845
10	0.5 to 0.36	0.5	11	85.92	14.08	12.18	5.31	82.51	.....	0.69	99.31	80.03	19.97	99.76	0.24	.....	.....	90.57	7.95	1.48	.....	1.72
11	0.36 to 0.28	0.5	11	85.70	14.30	15.10	3.17	81.73	.....	2.14	97.86	84.67	15.35	99.56	0.44	.....	.....	94.60	3.10	2.30	.....	1.59
	Classifier.																					
12	1st spigot	0.5	11	49.82	50.18	49.65	5.53	44.82	.....	0.74	99.26	86.15	13.85	99.71	0.29	.....	.....	98.21	1.53	0.26	.....	13.60
13	2d "	0.5	11	95.21	4.79	3.95	1.31	94.74	.....	1.38	98.62	66.05	33.95	99.64	0.36	.....	.....	81.35	10.72	7.93	.....	14.197
14	3 and 4 "	0.75	16.6	96.23	3.77	3.40	0.66	95.94	.....	1.15	98.85	65.57	34.43	99.80	0.20	.....	.....	88.95	5.98	5.07	.....	14.629
15	5 and 6 "	0.5	11	94.39	5.61	4.94	0.83	94.23	.....	1.65	98.35	71.10	28.90	99.45	0.55	.....	.....	86.66	4.29	9.05	.....	3.745
16	7, 8, 9 "	0.5	11	93.74	6.26	5.72	1.04	93.24	.....	1.16	98.84	83.75	16.25	99.54	0.46	.....	.....	90.58	2.90	6.52	.....	2.28
17	10,11,12 "	0.5	11	93.73	6.27	4.98	4.26	90.76	.....	0.38	99.62	83.72	16.28	99.34	0.66	.....	.....	79.38	10.69	9.93	.....	1.9

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TABLE VIII.—*Proportion of Quartz and Galena in Products.*

NOTE.—Concentrates sized and weighed. The weights are computed in tons and fractions of a ton on the 100-ton basis.

Sizes.		Run No. 1.	Run No. 2.	Run No. 3.	Run No. 4.	Run No. 5.
Through.	On.					
mm.	mm.	Tons.	Tons.	Tons.	Tons.	Tons.
.....	2.06	0.017	.....	.....	.....	0.009
2.06	1.63	0.059	.....	.....	.....	0.037
1.63	1.44	0.066	.....	.....	.....	0.054
1.44	1.27	0.112	.....	.....	.....	0.076
1.27	1.10	0.084	.....	.....	.....	0.049
1.10	0.97	0.120	0.003	.....	.....	0.079
0.97	0.84	0.174	0.012	.....	.....	0.104
0.84	0.68	0.206	0.027	.....	.....	0.128
0.68	0.57	0.245	0.048	.....	.....	0.138
0.57	0.45	0.506	0.280	0.225	.....	0.288
0.45	0.36	0.402	0.141	0.322	.....	0.201
0.36	0.28	0.445	0.465	1.000	0.227	0.303
0.28	0.24	0.306	0.304	0.386	0.081	0.179
0.24	0.20	0.375	0.370	0.586	0.760	0.230
0.20	0.15	0.307	0.324	0.483	0.997	0.218
0.15	0.12	0.209	0.313	0.483	0.958	0.213
0.12	0.10	0.110	0.207	0.271	0.397	0.082
0.10	0.08	0.189	0.235	0.404	0.871	0.229
0.08	0.00	0.182	0.244	0.507	1.052	0.191
Total tons.....		4.114	2.973	4.667	5.343	2.808

Analyses of the above products giving percentages of galena (PbS) and quartz (SiO_2).

Sizes.		Run No. 1.		Run No. 2.		Run No. 3.		Run No. 4.		Run No. 5.	
Thro.	On.	Galena.	Quartz.	Galena.	Quartz.	Galena.	Quartz.	Galena.	Quartz.	Galena.	Quartz.
mm.	mm.	Per Ct.	Per Ct.	Per Ct.	Per Ct.	Per Ct.	Per Ct.	Per Ct.	Per Ct.	Per Ct.	Per Ct.
.....	2.06	100.00	0.00	.....	.....	.....	.....	.....	.....	100.00	0.00
2.06	1.63	99.77	0.23	.....	.....	.....	.....	.....	.....	97.56	2.44
1.63	1.44	99.80	0.20	.....	.....	.....	.....	.....	.....	100.00	0.00
1.44	1.27	99.88	0.12	.....	.....	.....	.....	.....	.....	99.87	0.13
1.27	1.10	99.76	0.24	.....	.....	.....	.....	.....	.....	99.29	0.71
1.10	0.97	99.43	0.57	100.00	0.00	.....	.....	.....	.....	99.28	0.72
0.97	0.84	98.42	1.58	99.72	0.28	.....	.....	.....	.....	99.80	0.20
0.84	0.68	96.54	3.46	99.90	0.10	.....	.....	.....	.....	99.55	0.45
0.68	0.57	94.47	5.53	100.00	0.00	.....	.....	.....	.....	99.52	0.48
0.57	0.45	91.05	8.95	100.00	0.00	99.61	0.39	.....	.....	97.67	2.33
0.45	0.36	83.81	16.19	99.81	0.19	99.71	0.29	.....	.....	93.34	6.66
0.36	0.28	81.24	18.76	99.74	0.26	99.48	0.52	96.56	3.44	89.24	10.76
0.28	0.24	81.01	18.99	99.16	0.84	99.36	0.64	96.67	3.33	86.20	13.80
0.24	0.20	84.39	15.61	98.25	1.75	97.81	2.19	99.24	0.76	88.21	11.79
0.20	0.15	89.48	10.52	98.02	1.98	93.14	6.86	98.54	1.46	92.92	7.08
0.15	0.12	95.08	4.92	98.78	1.22	92.62	7.38	96.84	3.16	97.52	2.48
0.12	0.10	95.08	4.92	99.32	0.68	95.35	4.65	95.54	4.46	99.03	0.97
0.10	0.08	97.74	2.26	99.57	0.43	98.71	1.29	96.66	3.34	99.65	0.35
0.08	0.00	99.45	0.55	99.62	0.38	99.40	0.60	98.72	1.28	99.63	0.37
Total concentrates.		90.08	9.92	91.50	8.50	97.60	2.40	97.73	2.27	95.34	4.66

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TABLE IX.—*Proportion of Quartz and Galena in Products.*

NOTE.—Middlings sized and weighed and the weights computed in tons and fractions of a ton on the 100-ton basis.

Size. On.	Run No. 1.	Run No. 2.	Run No. 3.	Run No. 4.	Run No. 5.
mm.	Tons.	Tons.	Tons.	Tons.	Tons.
2.06	0.132	.....	.....	.....	0.206
1.63	0.795	.....	.....	.....	1.595
1.44	0.783	.....	.....	.....	2.134
1.27	1.202	.....	.....	.....	2.986
1.10	0.954	.....	.....	.....	2.438
0.97	1.056	0.292	.....	.....	3.178
0.84	1.552	1.330	.....	.....	4.793
0.68	2.021	0.329	.....	.....	5.113
0.57	1.865	1.350	.....	.....	4.323
0.45	3.112	3.203	2.171	.....	8.871
0.36	2.326	2.680	3.117	.....	3.144
0.28	2.526	3.945	5.955	0.402	3.655
0.24	0.848	1.436	4.805	1.684	1.079
0.20	1.230	2.932	3.698	4.729	0.969
0.15	0.648	1.551	4.362	4.713	0.581
0.12	0.207	0.893	0.481	3.660	0.223
0.10	0.075	0.299	0.418	2.296	0.050
0.08	0.061	0.286	0.601	2.216	0.054
0.00	0.093	0.224	0.373	1.328	0.146
Total tons.....	21.486	20.750	25.981	21.028	45.538

Analyses of the above products giving percentages of galena (PbS) and quartz (SiO_2).

Size.	Run No. 1.		Run No. 2.		Run No. 3.		Run No. 4.		Run No. 5.	
On.	Galena.	Quartz.	Galena.	Quartz.	Galena.	Quartz.	Galena.	Quartz.	Galena.	Quartz.
mm.	Per Ct.	Per Ct.	Per Ct.	Per Ct.	Per Ct.	Per Ct.	Per Ct.	Per Ct.	Per Ct.	Per Ct.
2.06	73.47	26.53	.....	.....	.....	.....	.....	.....	38.43	61.57
1.63	75.39	24.61	.....	.....	.....	.....	.....	.....	41.68	58.32
1.44	69.47	30.53	.....	.....	.....	.....	.....	.....	30.21	69.79
1.27	62.80	37.20	.....	.....	.....	.....	.....	.....	22.69	77.31
1.10	54.75	45.25	.....	.....	.....	.....	.....	.....	18.04	81.96
0.97	43.80	56.20	90.46	9.54	.....	.....	.....	.....	13.85	86.15
0.84	31.40	68.60	89.46	10.54	.....	.....	.....	.....	11.43	88.57
0.68	20.10	79.90	81.60	18.40	.....	.....	.....	.....	9.92	90.08
0.57	16.70	83.30	69.23	30.77	.....	.....	.....	.....	9.60	90.40
0.45	11.30	88.70	37.50	62.50	66.66	33.34	.....	.....	9.07	90.93
0.36	8.11	91.89	20.22	79.78	23.67	76.33	.....	.....	8.40	91.60
0.28	6.44	93.56	12.81	87.19	14.10	85.90	29.43	70.57	8.36	91.64
0.24	4.92	95.08	8.41	91.59	6.60	93.40	19.74	80.26	7.75	92.25
0.20	3.68	96.32	7.26	92.74	5.16	94.84	11.78	88.22	8.27	91.73
0.15	4.44	95.56	5.36	94.64	5.01	94.99	8.25	91.75	10.20	89.80
0.12	6.65	93.35	4.49	95.51	6.56	93.44	7.10	92.90	17.39	82.61
0.10	7.26	92.74	7.45	92.55	6.46	93.54	6.81	93.19	26.88	73.12
0.08	32.23	67.77	11.72	88.28	10.24	89.76	9.22	90.78	58.90	41.10
0.00	90.46	9.54	48.33	51.67	70.76	29.24	51.84	48.16	90.87	9.13
Total midd.	23.58	76.42	26.63	73.37	15.90	84.10	12.84	87.16	13.71	86.29

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TABLE X.—*Proportion of Quartz and Galena in Products.*

NOTE.—Tailings sized and weighed and the weights computed in tons and fractions of a ton on the 100-ton basis.

Size. On.	Run No. 1.	Run No. 2.	Run No. 3.	Run No. 4.	Run No. 5.
mm.	Tons	Tons.	Tons.	Tons.	Tons.
2.06	3.777	.....	.....	.....	2.003
1.63	14.713	.....	.....	.....	11.883
1.44	11.145	.....	.....	.....	9.065
1.27	8.085	.....	.....	.....	10.600
1.10	6.174	.....	.....	.....	4.294
0.97	7.322	11.860	.....	.....	4.452
0.84	5.191	16.217	.....	.....	1.489
0.68	3.350	8.821	.....	.....	0.682
0.57	2.078	5.789	.....	.....	0.405
0.45	2.501	11.639	20.635	.....	0.396
0.36	1.581	4.167	11.120	.....	0.162
0.28	1.056	3.315	10.790	0.400	0.456
0.24	0.523	0.915	2.829	2.019	0.303
0.20	0.633	2.341	3.277	12.345	0.680
0.15	0.739	1.860	3.574	11.948	1.075
0.12	1.139	1.781	4.308	9.015	0.507
0.10	0.624	1.067	1.113	6.118	0.614
0.08	0.948	1.461	2.965	9.778	0.878
0.00	1.277	1.670	2.202	8.864	0.742
Total tons....	72.856	72.903	62.813	60.487	50.686

Analyses of the above products giving percentages of galena (PbS) and quartz (SiO_2).

Size.	Run No. 1.		Run No. 2.		Run No. 3.		Run No. 4.		Run No. 5.	
On.	Galena.	Quartz.	Galena.	Quartz.	Galena.	Quartz.	Galena.	Quartz.	Galena.	Quartz.
mm.	Per Ct.	Per Ct.	Per Ct.	Per Ct.	Per Ct.	Per Ct.	Per Ct.	Per Ct.	Per Ct.	Per Ct.
2.06	0.40	99.60	.....	.....	.....	.....	.....	.....	0.00	100.00
1.63	0.40	99.60	.....	.....	.....	.....	.....	.....	0.00	100.00
1.44	0.50	99.50	.....	.....	.....	.....	.....	.....	0.00	100.00
1.27	0.20	99.80	.....	.....	.....	.....	.....	.....	0.05	99.95
1.10	0.01	99.99	.....	.....	.....	.....	.....	.....	0.02	99.98
0.97	0.02	99.98	0.02	99.98	.....	.....	.....	.....	0.02	99.98
0.84	0.00	100.00	0.01	99.99	.....	.....	.....	.....	0.03	99.97
0.68	0.00	100.00	0.00	100.00	.....	.....	.....	.....	0.01	99.99
0.57	0.00	100.00	0.00	100.00	.....	.....	.....	.....	0.08	99.92
0.45	0.00	100.00	0.00	100.00	0.00	100.00	.....	.....	0.05	99.95
0.36	0.08	99.92	0.12	99.88	0.16	99.84	.....	.....	0.68	99.32
0.28	0.12	99.88	0.16	99.84	0.64	99.36	2.95	97.05	0.80	99.20
0.24	0.69	99.31	0.26	99.74	2.03	97.97	0.41	99.59	0.74	99.26
0.20	1.24	98.76	0.80	99.20	1.19	98.81	0.69	99.31	0.82	99.18
0.15	1.45	98.55	1.40	98.60	1.49	98.51	1.00	99.00	0.76	99.24
0.12	1.82	98.18	1.89	98.11	1.65	98.35	1.26	98.74	1.16	98.84
0.10	2.88	97.12	1.69	98.31	1.60	98.40	1.76	98.24	1.71	98.29
0.08	3.43	96.57	2.60	97.40	1.51	98.49	1.49	98.51	3.00	97.00
0.00	17.50	82.50	17.85	82.15	8.55	91.45	4.28	95.72	12.17	87.83
Total tailin's	0.51	99.49	0.61	99.39	0.90	99.10	1.61	98.39	0.29	99.71

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sizes, but the quantity is so small that these losses are insignificant.

TABLE XI.—*Proportion of Quartz and Galena in Products.*

NOTE.—Slimes sized and weighed. The weights are computed in tons and parts of a ton, on the 100-ton basis.

Sizes.		Run No. 1.	Run No. 2.	Run No. 3.	Run No. 4.	Run No. 5.
Through.	On.					
mm.	mm.	Tons.	Tons.	Tons.	Tons.	Tons.
0.45	0.36	0.156	.....	.....	.....	.....
0.36	0.28	0.047	.....	.....	0.214	0.003
0.28	0.24	0.036	.....	.....	0.026	0.004
0.24	0.20	0.053	0.032	.....	0.044	0.007
0.20	0.15	0.063	0.072	0.207	0.091	0.027
0.15	0.12	0.052	0.062	0.078	0.138	0.037
0.12	0.10	0.068	0.149	0.241	1.515	0.038
0.10	0.08	0.134	0.446	1.662	1.268	0.244
0.08	0.00	0.935	2.613	4.351	9.846	0.606
Total tons.....		1.544	3.374	6.539	13.142	0.966

Analyses of the above products giving percentages of galena (PbS) and quartz (SiO_2).

Sizes.		Run No. 1.		Run No. 2.		Run No. 3.		Run No. 4.		Run No. 5.	
Thro.	On.	Galena.	Quartz.	Galena.	Quartz.	Galena.	Quartz.	Galena.	Quartz.	Galena.	Quartz.
mm.	mm.	Per Ct.	Per Ct.	Per Ct.	Per Ct.	Per Ct.	Per Ct.	Per Ct.	Per Ct.	Per Ct.	Per Ct.
0.45	0.36	0.00	100.00	.....	.....	.....	.....	.....	.....	.....	.....
0.36	0.28	10.20	89.80	.....	.....	.....	.....	48.54	51.46	18.30	81.70
0.28	0.24	10.00	90.00	.....	.....	.....	.....	54.55	45.45	15.40	84.60
0.24	0.20	10.81	89.19	14.04	85.96	.....	.....	40.22	59.78	11.80	88.20
0.20	0.15	12.58	87.42	7.94	92.06	45.21	54.79	37.16	62.84	8.10	91.90
0.15	0.12	19.00	81.00	7.26	92.74	21.21	78.79	28.96	71.04	6.50	93.50
0.12	0.10	24.28	75.72	7.19	92.81	17.53	82.47	9.37	90.63	5.00	95.00
0.10	0.08	27.04	72.96	5.03	94.97	8.53	91.47	6.87	93.13	6.40	93.60
0.08	0.00	28.31	71.69	10.72	89.28	11.53	88.47	8.76	91.24	19.40	80.60
Total slimes...		18.90	81.10	9.71	90.29	12.18	87.82	9.96	90.04	14.62	85.38

Commenting upon runs Nos. 6 to 11 (Table XII.), as compared with runs Nos. 1 to 5, we note immediately that the concentrates all the way through are almost pure galena with scarcely any quartz, and the tailings are almost pure quartz and scarcely any galena. The middlings, as remarked before, are so small in quantity that they affect the runs but little, and when we consider that they can go directly back on to the table in the continuous run, they do not affect the result at all. This set of runs, Nos. 6 to 11, therefore appears to distance runs Nos. 1 to 5 in the competition. There is really no comparison, since runs Nos. 1 to 5 are not in the same class with them.

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TABLE XII.—*Results of Runs Nos. 6 to 11 Computed to Full Size.*

NOTE.—The weights of products have been computed in tons and fractions thereof.

	Run No. 6.	Run No. 7.	Run No. 8.	Run No. 9.	Run No. 10.	Run No. 11.
	Tons.	Tons.	Tons.	Tons.	Tons.	Tons.
Concentrates.....	6.537	9.728	12.064	13.168	12.171	15.137
Middlings.....	1.647	1.901	1.694	1.602	5.310	3.170
Tailings.....	91.816	88.371	86.242	85.230	82.519	81.693
Slimes.....	0.000	0.000	0.000	0.000	0.000	0.000
Total.....	100.000	100.000	100.000	100.000	100.000	100.000

Analyses of the above products giving percentages of galena (PbS) and quartz (SiO_2).

	Run No. 6.		Run No. 7.		Run No. 8.		Run No. 9.		Run No. 10.		Run No. 11.	
	Galena.	Quartz.	Galena.	Quartz.	Galena.	Quartz.	Galena.	Quartz.	Galena.	Quartz.	Galena.	Quartz.
	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.
Concentrates.....	99.23	0.77	99.23	0.77	99.01	0.99	97.50	2.50	99.31	0.69	97.86	2.14
Middlings.....	59.67	40.33	23.10	76.90	17.00	83.00	22.60	77.40	19.97	80.08	15.35	84.65
Tailings.....	0.04	99.96	0.00	100.00	0.09	99.91	0.35	99.65	0.24	99.76	0.44	99.56
Slimes.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....

Comparing the concentrates of runs Nos. 12 to 17 (Table XIII.), we see in the first place an enormous heaping-up of concentrates in run No. 12. The weight, 49 tons, is more than double the weight of all the other five runs in this set put together. This heaping-up of the great quantity of concentrates on the table which treats the first spigot of a classifier is one of the prominent features of the use of a classifier in preparation for feeding Wilfley tables. If we look at the total percentage of galena and quartz in the concentrates of run No. 12 we see that they contain 99.26 per cent. of galena and 0.74 per cent. of quartz. This makes an extremely good showing, and one which bids for favorable consideration of the classifier set.

Looking at the later spigots, that is to say, runs Nos. 13 to 17, inclusive, in the analyses, we see that the percentage of quartz looks high in the coarser sizes. This would seem a serious disadvantage if it were not for the fact that these products which have the high percentages of quartz are so small in quantity that the quartz cuts scarcely any figure in the final percentage of quartz in the concentrates. Altogether this set

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TABLE XIII.—*Results of Runs Nos.* 12 *to* 17 *Computed to Full Size.*

NOTE.—The concentrates have been sized and weighed and the weights computed in tons and fractions thereof on the 100-ton basis.

Sizes.		Run No. 12.	Run No. 13.	Run No. 14.	Run No. 15.	Run No. 16.	Run No. 17.
Through.	On.						
mm.	mm.	Tons.	Tons.	Tons.	Tons.	Tons.	Tons.
.....	2.06	0.128	.....	.....	.....	.....	.....
2.06	1.63	4.869	0.001	0.0003	.....	.....	.....
1.63	1.44	6.331	0.001	0.0003	.....	.....	.....
1.44	1.27	7.861	0.002	0.0003	.....	.....	.....
1.27	1.10	4.455	0.002	0.0002	0.0005	.....	.....
1.10	0.96	4.093	0.002	0.0003	0.0003	.....	.....
0.96	0.84	5.837	0.011	0.001	0.0003	0.0005	.....
0.84	0.68	3.674	0.036	0.002	0.0005	0.0005	0.0004
0.68	0.57	3.615	0.124	0.006	0.001	0.001	0.0004
0.57	0.45	4.347	0.730	0.054	0.005	0.004	0.0023
0.45	0.36	2.063	1.208	0.174	0.012	0.007	0.004
0.36	0.28	1.232	0.687	0.656	0.060	0.049	0.015
0.28	0.24	0.445	0.384	0.415	0.110	0.077	0.015
0.24	0.20	0.330	0.393	0.697	0.332	0.138	0.057
0.20	0.15	0.152	0.223	0.701	1.176	0.220	0.108
0.15	0.12	0.088	0.084	0.393	1.321	0.634	0.086
0.12	0.10	0.029	0.021	0.143	0.551	0.748	0.109
0.10	0.08	0.055	0.029	0.104	1.057	2.300	0.934
0.08	0.00	0.045	0.012	0.052	0.315	1.539	3.650
Total tons.....		49.649	3.950	3.398	4.940	5.717	4.980

Analyses of the above products giving percentages of galena (PbS) and quartz (SiO_2).

Size.	Run No. 12.		Run No. 13.		Run No. 14.		Run No. 15.		Run No. 16.		Run No. 17.	
On.	Galena.	Quartz.	Galena.	Quartz.	Galena.	Quartz.	Galena.	Quartz.	Galena.	Quartz.	Galena.	Quartz.
mm.	Per Ct.	Per Ct.	Per Ct.	Per Ct.	Per Ct.	Per Ct.	Per Ct.	Per Ct.	Per Ct.	Per Ct.	Per Ct.	Per Ct.
2.06	100.00	0.00	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....
1.63	99.96	0.04	100.00	0.00	60.00	40.00	.....	.....	.....	.....	.....	.....
1.44	99.96	0.04	100.00	0.00	75.00	25.00	.....	.....	.....	.....	.....	.....
1.27	99.95	0 05	93.10	6.90	75.00	25.00	.....	.....	.....	.....	.....	.....
1.10	99.84	0.16	91.45	8.55	85.71	14.29	0.00	100.00	.....	.....	.....	.....
0.96	99.69	0.31	89.69	10.31	80.00	20.00	87.50	12.50	.....	.....	.....	.....
0.84	99.45	0.55	95.63	4.37	98.13	1.87	85.72	14.28	0.00	100.00	.....	.....
0.68	99.03	0.97	96.15	3.85	97.58	2.42	50.00	50.00	63.63	36.37	50.00	50.00
0.57	98.76	1.24	97.49	2.51	97.75	2.25	50.00	50.00	88.24	11.76	33.33	66.67
0.45	97.80	2.20	98.01	1.99	96.06	3.94	73.36	26.64	88.78	11.22	93.80	6.20
0.36	97.05	2.95	98.80	1.20	96.58	3.42	20.50	79.50	92.53	7.47	96.52	3.48
0.28	96.96	3.04	98.24	1.76	98.02	1.98	95.84	4.16	93.54	6.46	98.20	1.80
0.24	96.85	3.15	98.96	1.04	98.57	1.43	97.46	2.54	97.44	2.56	98.58	1.42
0.20	97.51	2.49	99.35	0.65	99.05	0.95	98.93	1.07	96.37	3.63	99.36	0.64
0.15	97.51	2.49	99.58	0.42	99.42	0.58	99.46	0.54	96.41	3.59	99.19	0.81
0.12	98.88	1.12	99.81	0.19	99.80	0.20	99.76	0.24	97.14	2.86	97.56	2.44
0.10	97.77	2.23	99.82	0.18	99.88	0.12	99.83	0.17	98.37	1.63	96.92	3.08
0.08	97.71	2.29	99.83	0.17	99.86	0.14	99.86	0.14	99.31	0.69	99.30	0.70
0.00	96.86	3.14	99.64	0.36	99.88	0.12	99.84	0.16	99.84	0.16	99.86	0.14
Total concn'ts.	99.26	0.74	98.62	1.38	98.85	1.15	98.35	1.65	98.84	1.16	99.62	0.38

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TABLE XIV.—*Results of Runs Nos.* 12 *to* 17 *Computed to Full Size.*

NOTE.—The middlings have been sized and weighed and the weights computed in tons and fractions thereof on the 100-ton basis.

Sizes.		Run No. 12.	Run No. 13.	Run No. 14.	Run No. 15.	Run No. 16.	Run No. 17.
Through.	On.						
mm.	mm.	Tons.	Tons.	Tons.	Tons.	Tons.	Tons.
.....	2.49	0.005	.....	.....	.....	.....	.....
2.49	2.06	0.061	0.001	.....	.....	.....	.....
2.06	1.63	0.788	0.015	0.002	.....	.....	.....
1.63	1.44	0.602	0.026	0.001	.....	.....	.....
1.44	1.27	0.951	0.037	0.002	.....	.....	.....
1.27	1.10	0.676	0.014	0.002	.....	.....	.....
1.10	0.96	0.605	0.018	0.002	0.0003	.....	.....
0.96	0.84	0.758	0.078	0.005	0.0003	0.001	.....
0.84	0.68	0. 73	0.120	0.011	0.002	0.001	0.001
0.68	0.57	0.263	0.194	0.033	0.003	0.003	0.001
0.57	0.45	0.234	0.494	0.092	0.020	0.011	0.003
0.45	0.36	0.042	0.188	0.133	0.031	0.025	0.004
0.36	0.28	0.030	0.089	0.236	0.139	0.064	0.075
0.28	0.24	0.015	0.014	0.048	0.093	0.070	0.080
0.24	0.20	0.011	0.011	0.068	0.250	0.213	0.281
0.20	0.15	0.004	0.003	0.020	0.146	0.299	0.809
0.15	0.12	0.004	0.002	0.005	0.076	0.188	1.013
0.12	0.10	0.001	0.001	0.001	0.015	0.051	0.531
0.10	0.08	0.004	0.001	0.001	0.028	0.060	0.909
0.08	0.00	0.007	0.001	0.001	0.025	0.057	0.555
Total tons.....		5.534	1.307	0.663	0.828	1.043	4.262

Analyses of the above products giving percentages of galena (PbS) and quartz (SiO_2).

Size.	Run No. 12.		Run No. 13.		Run No. 14.		Run No. 15.		Run No. 16.		Run No. 17.	
On.	Galena.	Quartz.	Galena.	Quartz.	Galena.	Quartz.	Galena.	Quartz.	Galena.	Quartz.	Galena.	Quartz.
mm.	Per Ct.	Per Ct.	Per Ct.	Per Ct.	Per Ct.	Per Ct.	Per Ct.	Per Ct.	Per Ct.	Per Ct.	Per Ct.	Per Ct.
2.49	0.00	100.00	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....
2.06	22.34	77.66	0.00	100.00	.....	.....	.....	.....	.....	.....	.....	.....
1.63	31.80	68.20	8.84	91.16	68.18	31.82	.....	.....	.....	.....	.....	.....
1.44	25.40	74.60	10.87	89.13	55.55	44.45	.....	.....	.....	.....	.....	.....
1.27	17.70	82.30	21.33	78.67	31.82	68.18	.....	.....	.....	.....	.....	.....
1.10	7.93	92.07	28.29	71.71	41.40	58.60	.....	.....	.....	.....	.....	.....
0.96	6.73	93.27	39.39	60.61	33.33	66.67	16.67	83.33	.....	.....	.....	.....
0.84	4.04	95.96	38.92	61.08	44.45	55.55	16.67	83.33	66.67	33.33	.....	.....
0.68	3.10	96.90	29.42	70.58	42.44	57.56	59.45	40.55	23.08	76.92	64.29	35.71
0.57	2.43	97.57	25.14	74.86	37.50	62.50	61.54	38.46	66.67	33.33	52.39	47.61
0.45	3.56	96.44	31.00	69.00	38.80	61.20	62.27	37.73	79.58	20.42	75.52	24.48
0.36	5.58	94.42	47.93	52.07	38.58	61.42	59.76	40.24	47.72	52.28	83.34	16.66
0.28	9.57	90.43	49.53	50.47	32.44	67.56	33.33	66.67	28.51	71.49	28.17	71.83
0.24	16.15	83.85	46.20	53.80	28.12	71.88	18.51	81.49	11.15	88.85	10.67	89.33
0.20	22.80	77.20	52.99	47.01	26.72	73.28	18.45	81.55	5.86	94.14	4.60	95.40
0.15	27.33	72.67	71.90	28.10	32.77	67.23	16.93	83.07	4.91	95.09	2.25	97.75
0.12	95.98	4.02	88.22	11.78	42.00	58.00	23.79	76.21	6.27	93.73	3.54	96.46
0.10	96.91	3.09	92.04	7.96	60.78	39.22	41.44	58.56	11.32	88.68	5.10	94.90
0.08	97.92	2.08	91.95	8.05	74.16	25.84	79.81	20.19	40.12	59.88	11.12	88.88
0.00	98.06	1.94	96.40	3.60	6.32	93.68	98.07	1.93	90.42	9.58	82.98	17.02
Total midd's	13.85	86.15	33.95	66.05	34.43	65.57	28.90	71.10	16.25	83.75	16.28	83.72

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of concentrates is so good that it would be accepted by any mill, even the Missouri mills, where such clean concentration is required.

Looking at the middlings (Table XIV.), we see that the quantity is extremely small, and can be made to disappear in a continuous run by feeding them back on to the table without harming either the concentrates or the tailings. They have, however, some very interesting features that are worthy of note. The galena in the 12th run runs high in the coarse and in the fine, and very low in the middle sizes, there being a great heaping-up of quartz in this part. This same point is true in the 16th and 17th runs to a very marked degree. It is again true to a less marked degree in the 14th and 15th runs, and it appears not to be true at all in the 13th run.

Looking at the tailings (Table XV.), we see that run No. 12 stands out pre-eminent, having only 0.29 per cent. of galena in the whole tailings, and the tailings of runs Nos. 13 to 17 are very low in galena, and would probably pass in any concentrating establishment.

We have one feature here which does not and cannot happen in sized runs, Nos. 6 to 11—viz., the tailings get richer in galena down to the finer sizes; but when we look at the tonnage we find that there is scarcely any weight of material down in those sizes, and therefore this loss is not serious and does not bring up the percentage of galena in the final tailings to a serious extent.

Fig. 4 is an ideal sketch of what happens at the discharging-corner of a Wilfley table. Running from coarse on the lower edge to fine on the upper, *A*, *B*, *C*, *D*, *E*, *F*, *G*, and *H* represent the different sizes of galena. It appears that they arrange themselves approximately according to this order on the Wilfley table. In like manner, the quartz-grains arrange themselves approximately in order of size, beginning at the lower edge with the largest grade and running smaller and smaller upwards, as indicated by the letters *I*, *J*, *K*, *L*, *M*, *N*, *O*, and *P*. The slimes at once take off the galena (*H*), and the quartz (*P*). These finest of all grains have not sufficient weight to hold them up to the upper edge, where mathematical logic would place them. They therefore go into the slimes. The next grade, *G* (galena), and *O* (quartz), are not fine enough to

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TABLE XV.—*Results of Runs Nos. 12 to 17 Computed to Full Size.*

NOTE.—The tailings have been sized and weighed and the weights computed in tons and fractions thereof on the 100-ton basis.

Size. On.	Run No. 12.	Run No. 13.	Run No. 14.	Run No. 15.	Run No. 16.	Run No. 17.
mm.	Tons.	Tons.	Tons.	Tons.	Tons.	Tons.
2.49	0.042	0.043	.....	.....	.....	.....
2.06	1.762	1.336	0.056	0.027	.....	.....
1.63	18.040	18.429	2.083	0.023	.....	0.016
1.44	10.232	21.674	4.434	0.017	0.016	0.011
1.27	8.176	20.207	11.129	0.030	0.016	0.011
1.10	2.828	11.066	9.834	0.071	0.016	0.011
0.96	1.590	7.877	8.167	0.185	0.021	0.022
0.84	1.457	7.835	13.670	0.780	0.026	0.016
0.68	0.281	2.676	13.131	1.843	0.047	0.033
0.57	0.178	2.098	11.080	7.316	0.163	0.044
0.45	0.103	1.122	14.410	21.851	2.381	1.945
0.36	0.021	0.121	4.532	20.371	5.709	5.423
0.28	0.019	0.097	2.604	30.530	34.072	31.480
0.24	0.015	0.038	0.437	5.486	17.393	17.182
0.20	0.017	0.024	0.220	4.473	18.103	19.917
0.15	0.012	0.019	0.087	0.790	10.150	10.175
0.12	0.015	0.014	0.033	0.240	3.129	2.845
0.10	0.002	0.009	0.010	0.084	1.249	0.911
0.08	0.010	0.029	0.010	0.071	0.580	0.544
0.00	0.017	0.029	0.012	0.044	0.169	0.172
Total tons.....	44.817	94.743	95.939	94.232	93.240	90.758

Analyses of the above products giving percentages of galena (PbS) and quartz (SiO_2).

Size.	Run No. 12.		Run No. 13.		Run No. 14.		Run No. 15.		Run No. 16.		Run No. 17.	
On.	Galena.	Quartz.	Galena.	Quartz.	Galena.	Quartz.	Galena.	Quartz.	Galena.	Quartz.	Galena.	Quartz.
mm.	Per Ct.	Per Ct.	Per Ct.	Per Ct.	Per Ct.	Per Ct.	Per Ct.	Per Ct.	Per Ct.	Per Ct.	Per Ct.	Per Ct.
2.49	0.00	100.00	0.00	100.00	.....	.....	.....	.....	.....	.....	.....	.....
2.06	0.00	100.00	0.00	100.00	0.00	100.00	0.00	100.00	.....	.....	.....	.....
1.63	0.00	100.00	0.08	99.92	0.00	100.00	0.00	100.00	.....	.....	.....	.....
1.44	0.52	99.48	0.07	99.93	0.09	99.91	0.00	100.00	0.00	100.00	0.00	100.00
1.27	0.20	99.80	0.48	99.52	0.06	99.94	0.00	100.00	0.00	100.00	.....	.....
1.10	0.38	99.62	0.09	99.91	0.06	99.94	0.00	100.00	0.00	100.00	.....	.....
0.96	0.67	99.33	0.18	99.82	0.03	99.97	0.00	100.00	0.00	100.00	.....	.....
0.84	0.74	99.26	0.18	99.82	0.03	99.97	0.00	100.00	0.00	100.00	.....	.....
0.68	0.76	99.24	0.54	99.46	0.02	99.98	0.00	100.00	1.10	98.90	.....	.....
0.57	1.20	98.80	1.16	98.84	0.02	99.98	0.00	100.00	0.32	99.68	0.00	100.00
0.45	1.03	98.97	6.04	93.96	0.10	99.90	0.17	99.83	0.86	99.14	1.36	98.64
0.36	1.28	98.72	16.00	84.00	0.84	99.16	0.23	99.77	0.45	99.55	0.69	99.31
0.28	7.08	92.92	33.05	66.95	2.08	97.92	0.98	99.02	0.17	99.83	0.34	99.66
0.24	9.22	90.78	27.60	72.40	3.88	96.12	0.27	99.73	0.30	99.70	0.58	99.42
0.20	5.64	94.36	19.00	81.00	5.21	94.79	0.43	99.57	0.35	99.65	0.55	99.45
0.15	5.90	94.10	13.32	86.68	7.73	92.27	3.30	96.70	0.44	99.56	0.54	99.46
0.12	7.50	92.50	15.30	84.70	9.59	90.41	8.97	91.03	0.87	99.13	1.01	98.99
0.10	5.40	94.60	40.40	59.60	72.79	27.21	17.32	82.68	1.83	98.17	2.07	97.93
0.08	28.38	71.62	55.55	44.45	21.43	78.57	26.55	73.45	7.04	92.96	7.38	92.62
0.00	85.93	14.07	87.00	13.00	68.68	31.32	68.32	31.68	42.36	57.64	44.28	55.72
Total tail'gs.	0.29	99.71	0.36	99.64	0.20	99.80	0.55	99.45	0.46	99.54	0.66	99.34

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go into the slimes nor coarse enough to stand up against the water-current in the position shown in the sketch. These grains are found, therefore, sprinkled through the concentrates, middlings and tailings. See the heaping-up of galena in the small sizes in Tables VIII., IX. and X.

Having laid out our argument in this way, it now remains for us to compare by means of this diagram runs Nos. 1 to 5,

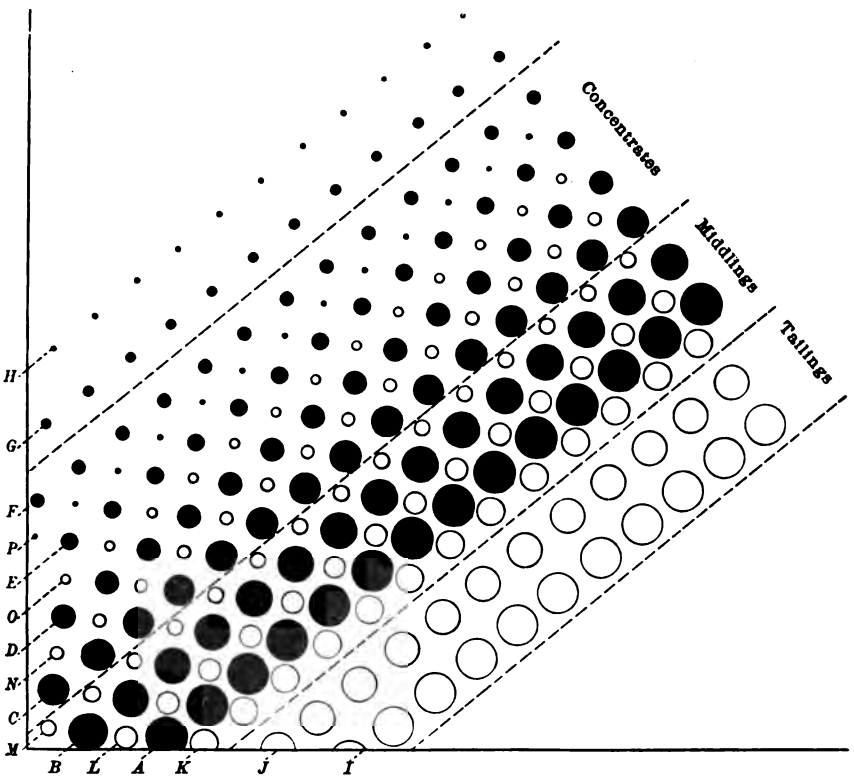

FIG. 4.—IDEAL SKETCH OF THE ARRANGEMENT OF GRAINS BY A WILFLEY TABLE.

6 to 11, and 12 to 17, and to see why it is that runs Nos. 6 to 11 and Nos. 12 to 17 are so much better than runs Nos. 1 to 5. Runs Nos. 1 to 5 take the products just as they are shown in Fig. 4 and give galena, *C*, *D*, *E*, *F*, in the concentrates contaminated by quartz, (*N*). See the heaping-up of the quartz a little below the middle size in Table VIII., and the middlings that give quartz, *K*, *L*, and *M*, contaminated by galena, *A*, *B*,

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C, and *G*. See the heaping-up of galena among the large grains and among the small grains in Table IX., and the tailings have in them the quartz, *I*, *J*, and *O*, contaminated by galena, *G*. See the heaping-up of galena in the fine sizes in Table X. Runs Nos. 6 to 11, on the other hand, have put together on the coarse table, quartz, *I*, and galena, *A*, which have nothing whatever to do with one another (see Table XII.), and therefore make almost 100 per cent. of galena in the concentrates, and almost 100 per cent. of quartz in the tailings. The little accidental middling-product, simply being the dividing-line between the two products, goes back on the table and disappears. On the second table we treat quartz, *J*, and galena, *B*, with the same result. On the third table we treat quartz, *K*, and galena, *C*, with the same result. On the fourth table quartz, *L*, and galena, *D*; on the fifth table quartz, *M*, and galena, *E*; on the sixth table quartz, *N*, and galena, *F*. There seems no reason logically why these should not turn out 100 per cent. of galena in the concentrates and 100 per cent. of quartz in the tailings. The probable reason why we did not obtain those figures was that the accidental flat scales and the fine abrasions of galena went where they should not.

Going to the third set of runs, Nos. 12 to 17, we need to bring in an ideal picture of the products of a classifier by means of Fig. 5. Suppose, for example, that we drop into a tall tube of water grains of quartz ranging from our maximum size down to zero, and grains of galena in the same way, and that these grains are of approximately the same shape, then the rate of settling of these grains may be stated in the following terms: the larger grains of a single mineral will settle faster than the smaller grains; and when we compare the two gravities of quartz and galena, the higher gravity will settle faster than the lower gravity for the same size. So definite is this law that if we look for equal-settling particles, we shall find that the grain of quartz which is equal-settling with the grain of galena is about 3 or 4 times the diameter of the grain of galena. See settling-ratios in Table III. We may, therefore, construct the ideal diagram, Fig. 5, and we can draw a set of horizontal lines across it, putting the equal-settling grains together, ranging from the heavier grains of the first spigot in the lower part of the diagram up to the lighter grains of the

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finer spigot at the upper part of the diagram. We see then that spigot 1 contains a large amount of galena ranging from the coarsest size down to one-quarter the diameter of the coarsest quartz, and that the quartz is almost all in the coarse sizes. See spigot 1, Table III. This is exactly what we found in our run No. 12. See the heaping-up of galena in the large sizes in Table XIII.

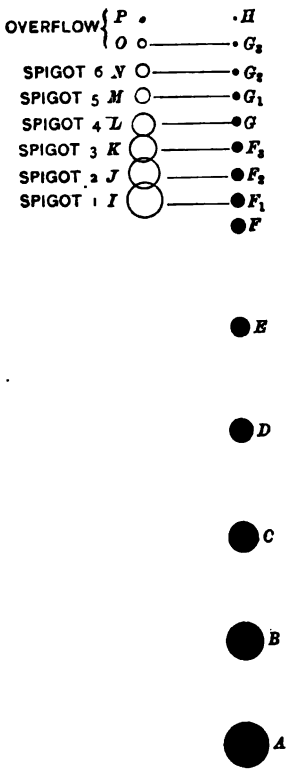

FIG. 5.—IDEAL SKETCH OF CLASSIFIER-PRODUCT.

Spigot 2 has small galena and large quartz, but both are a little smaller than those in spigot 1. Spigot 3, again, has small galena and larger quartz, but a little smaller than spigot 2, and so on up the scale with spigot 4, spigot 5 and spigot 6. See the heaping-up of galena in smaller sizes, Table XIII., and of quartz in larger sizes in Table XV.

Looking at our diagram, Fig. 5, to see what will happen when these several spigots are put upon the table, we shall

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find that run No. 12 receives galena, *A*, *B*, *C*, *D*, *E*, *F*, and F_1, and quartz, *I*. Logically these have nothing to do with one another, and therefore should make for perfect separation. Spigot 2 fed in run No. 13 would have quartz, *J*, and galena, F_2. Spigot 3 would have quartz, *K*, and galena, F_3, and so on. Spigots 4, 5, and 6 could work their way up, having quartz always larger, and therefore belonging at a lower place on the table, and galena of smaller diameter belonging at a higher place on the table, making for clean separation of concentrates and tailings, with a middling product that can go directly back on the table and disappear. In proof of this, see points of heaping-up of galena and quartz in Tables XIII. and XV.

In the light of Fig. 4, comparing runs 12 to 17 with runs 6 to 11 we see that the natural lines for quartz and galena are farther apart for the classified products than for the sized products. For example, in run 12 the galena lines *A* to *F* average farther from the quartz *I* than does the galena *A* of run 6. Again, in run 13 the galena *F* is farther from quartz *J* than is the galena *B* from quartz *J* in run 7. In like manner we may compare classified runs 14, 15, 16, and 17 with sized runs 8, 9, 10, and 11.

This demonstrates that with perfect classification the work will be better done on the Wilfley table than with sizing, and it also shows that with much middle weight mineral or included grains a good classifier will probably be more efficient than screens.

Conclusions.

1. The natural product as feed for a Wilfley table is completely outclassed and surpassed by sized-product feed and by classifier-product feed.

2. While the sized-product feed, as shown in Table VII., appears to have done better work than the classifier-product feed, if we give full weight to the great performance of run No. 12, we can agree that this has fully offset the slight falling-off of runs Nos. 13 to 17, and that the classifier-feed work is fully up to the sized-feed work on the Wilfley table, and with a perfect classifier the work will be better done than with screens.

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Coal-Briquetting in the United States.*

BY EDWARD W. PARKER, WASHINGTON, D. C.

(Toronto Meeting, July, 1907.)

NOTE.—The material from which this paper has been prepared was collected for the *U. S. Geological Survey Bulletin*, No. 316, Contributions to Economic Geology, 1906, and appears also, though in somewhat more extended form without illustrations, in that publication.

ALTHOUGH the briquetting of coals and lignites has been carried on for many years in Europe, and has reached a particularly high state of development in France, Belgium, and Germany, it has made comparatively little progress in the United States. The causes for the backwardness of the United States in this regard are several, and first among them has been the abundant supply of cheap raw fuel with which the manufactured article has to compete. With our millions of acres of coal-productive territory, from which the product can in most cases be cheaply extracted, it has appeared in many districts more economical to waste the slack or culm, which constitutes a considerable percentage of the product, than to attempt to save it at the additional expense required for briquetting. It is for this reason that the view in all sections of the anthracite-region of Pennsylvania is marred by the unsightly culm-banks which encumber the ground, and that in some of the bituminous-districts one sees huge piles of unmarketable slack allowed to burn up in order to get rid of them. When the coal is of a coking quality, or when the slack can be used for steaming-purposes, these losses are not sustained, but many thousands of tons of what might be converted into usable fuel have been wasted every year simply because of the increased expense involved in its preparation.

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The rational development of the briquetting-industry has also been retarded by the attempts to exploit patented or secret processes for which all kinds of extravagant claims have been made, but which have almost invariably proved expensive and unprofitable, and the investment of capital in enterprises of this character has been discouraged accordingly. The Patent Office records teem with patents issued on all sorts of inventions relating to binders, many of which are as fanciful as the idea of perpetual motion.

Another reason for the failure to build up a briquetting-industry in the anthracite-region of Pennsylvania, where the best opportunity for its development is offered, has been the opposition shown by some of the operators to the introduction of a manufactured domestic fuel which would come into competition with the prepared sizes of anthracite. And such an opposition is natural. The competition of bituminous coal has almost entirely shut out anthracite as a steam-fuel. The use of coke has supplanted the use of anthracite for iron-making, and the use, for domestic purposes, of coke and gas made from bituminous coal is growing. Owing to the greater depths to which the mining of anthracite is being carried, the thinner and less favorably located beds which are being worked, and to the increasing tendency in labor-cost, the mining and preparing of anthracite are constantly becoming more expensive on one hand, while competition is becoming more keen on the other. A certain rate of production must be kept up for the protection of the properties themselves, and when all these conditions are considered, the unfavorable attitude on the part of the operators toward further competition is at least realizable.

Still another reason which has been assigned, rightly or wrongly, for our halting progress in fuel-briquetting has been the lack of assurance of a regular supply of coal-tar pitch at reasonably low prices. Out of the many, and sometimes costly, attempts that have marked the incubating period of briquetting development has grown the knowledge that coal-tar pitch must be relied upon to supply, in the eastern States at least, all, or the greater part, of the binding-material. In California, Arizona, and other parts of the far West, asphaltic pitch, the residual product from the refining of the heavy asphalt-base petroleums of that region, has been, and is now, successfully

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used in recently constructed briquetting-plants. But in the East, coal-tar pitch is the base of the economically successful cementing-material. The extended investigations carried on at the U. S. Geological Survey coal-testing plant at St. Louis bear out this statement.[1] These investigations included experiments with all kinds of organic and inorganic binders, embracing, besides coal-tar pitch, such materials as rosin, sugar-house refuse, molasses, acid-sludge, quick-lime, and various mixtures. The results show that coal-tar and asphaltic pitch are the only really successful binders. Any materials used with them must possess above all others the essential virtue of cheapness.

But while it is claimed that the briquetting-industry has been held back by the lack of assurance of a steady supply of coal-tar pitch, it also happens that one of the reasons assigned for the comparatively slow development of the by-product coking-ovens in the United States in the last few years is the lack of a profitable demand for coal-tar, one of the principal by-products of the retort coke-ovens. It is well known that the demand for creosoting-oils to be used for the preservation of ties, bridge-timbers, etc., by the railroad-companies is far beyond the present domestic production of that coal-tar product, and the statistics compiled by the Bureau of Statistics, Department of Commerce and Labor, show that our imports of the chemical products of coal-tar exceed $10,000,000 in value yearly. To the ordinary observer it would appear that the conditions here presented afford an opportunity for the organization of a community of interests which would prove profitable to the projectors and beneficial to the general public. The constantly increasing expense involved in the mining and preparation of anthracite coal is making that commodity slowly but surely more and more of a luxury, and manufactured fuel which will take the place of anthracite for domestic use, particularly among consumers of moderate means, appears to be needed. This is especially true in the northeastern section of the United States.

Two of the briquetting-plants recently constructed, and which are discussed in more detail in the following pages, in-

[1] *United States Geological Survey Bulletins* Nos. 261 and 290 (1905), and *Professional Paper* No. 48 (1906).

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dicate somewhat of a "getting together" of the coal-tar producing and the briquetting-interests. These are the plants of the United Gas Improvement Co., at Point Breeze, Philadelphia, and of the Semet-Solvay Co., at Del Ray, Mich. Both companies are producers of coal-tar, and the plants have been constructed for the purpose of briquetting mixtures of anthracite-culm and coke-breeze.

It appears now, moreover, that the period of failure and discouragement has passed, and that the manufacture and use of briquetted fuel is being placed upon a substantial footing. The first successful plant in the United States of which I have any definite knowledge was one built at Stockton, Cal., a few years ago by the San Francisco & San Joaquin Coal Co. This plant, unfortunately, was entirely destroyed by fire in 1905, and the plans for its reconstruction, this time at San Francisco, were interrupted by the earthquake and fire which destroyed a large portion of that city in April, 1906.

During the past two years a number of briquetting-plants have been constructed, and as complete descriptions of them as it has been possible to obtain are given in the subsequent pages. Some of them have been put in operation since Jan. 1, 1907.

NEW YORK, N. Y.

New Jersey Briquetting Co.—During 1904 and 1905 the New Jersey Briquetting Co. of New York constructed at the foot of Washington Street, in Brooklyn, a plant for exploiting the briquetting-process of the Zwoyer Fuel Co. This plant was intended to be operated in connection with a coal-yard on Adams Street, but during the construction of the piers and anchorages for the new Manhattan bridge, the company was prohibited from operating the tramway from the coal-yard to the plant. This naturally interfered with the operations of the plant, and as extensive storage-capacity, either for raw material or for the product, had not been provided for, the work already done has been accomplished under much disadvantage. The prohibition put upon the tramway, and the lack of dock-facilities for loading and unloading material, have crippled the plant to such an extent that what was supposed to be an excellent location has turned out to be an unfortunate one, and the present methods of receiving and handling the material make the

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operations too expensive for successful competition of the briquettes with raw fuel. As a result of these unfortunate conditions it is proposed to remove the plant to a site better adapted for the receiving, storage, and shipment of material. The officials of both the New Jersey Briquetting Co. and the Zwoyer Fuel Co. are entirely satisfied with the experimental results, and are confident of making the enterprise a commercial success.

A description of the plant in Brooklyn has already been published,[2] and the following notes are partly abstracted therefrom and partly furnished by Mr. Virgil H. Hewes, Treasurer of the Zwoyer Fuel Co.

Prior to the construction of the plant in Brooklyn, the Zwoyer Fuel Co. had built a small experimental plant in Jersey City, N. J., but while this was of sufficient capacity for this purpose, it was not large enough to be operated as a commercial undertaking, and was abandoned.

It may be stated here that after a considerable expenditure of time and money in experimenting with different kinds of binders, coal-tar pitch was decided upon as best suited to the work, a decision which has been generally reached in the eastern States, as asphaltic pitch has been adopted in the far West, where that article is cheaply obtained. During the progress of the experimental work, about 200 tons of briquettes were made with a binder composed of 6.25 per cent. of rosin and oil, 1.5 per cent. of flour and water, and from 6 to 10 per cent. of bituminous coal, the body of the briquette being anthracite-dust. About 900 tons of briquettes were made, using from 5 to 7 per cent. of rosin and oil and 10 per cent. of bituminous coal; 400 tons were made with from 5 to 7 per cent. of wood-pitch and 10 per cent. of bituminous coal, and 1,500 tons were made with from 6 to 7 per cent. of coal-tar pitch alone. In applying the binder during the last three experiments an atomizer was used.

The plant in Brooklyn has a capacity of 10 tons per hour, and was built for the purposes of demonstration. During the winter and spring of 1905–6 about 3,000 tons of anthracite-briquettes were made and sold. The price received was $5 per ton of 2,000 lb. at the plant, $5.50 per ton delivered, and

[2] *Iron Age*, vol. lxxvii., pp. 1330 to 1333 (1906).

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$6.60 per ton in bags of 100 lb. each. These prices were $0.50 below the prices of the domestic sizes of anthracite.

A plan of the building,[3] which is nearly triangular in outline, is shown in Fig. 1, which gives also the general arrangement of the machinery.

The anthracite-dust is received on Washington Street at the end of a screw-conveyor, which carries it to the foot of an elevator, where it is lifted to the top of the plant and is then spouted to a screen located over the dust-bin. The coarser material is spouted either to the boiler-room or to an over-size bin in the rear of the dust-bin, and from here fed into a crusher

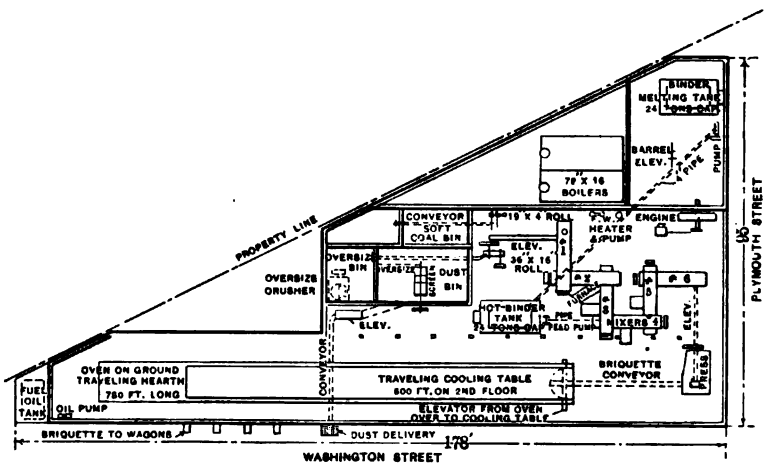

FIG. 1.—PLAN OF THE PLANT OF THE NEW JERSEY BRIQUETTING CO., BROOKLYN, N. Y.

and then passes to the foot of the dust-elevator, where it is again carried to the screen. The dust is drawn from the dust-bin by a conveyor driven from a variable-speed countershaft, and is fed to the 16-in. by 36-in. roll-crusher. It then passes to an elevator which carries it to the mixers. After passing through mixers Nos. 1 to 6 it is carried to the second floor, when it falls into the press-hopper.

From the press the briquettes are carried by a belt-conveyor to the baking-oven (when smokeless briquettes are wanted), and are then elevated to and distributed upon the cooling-table, which is located on the second floor. After cooling, the

[3] *Iron Age*, vol. lxxvii., p. 1330 (1906).

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briquettes are run into chutes and loaded into wagons for delivering, or are stored. In New York the briquettes sold readily when not baked.

On one side of the dust-bin a soft-coal bin was built from which soft coal was fed into a 19-in. by 4-in. roll-crusher and

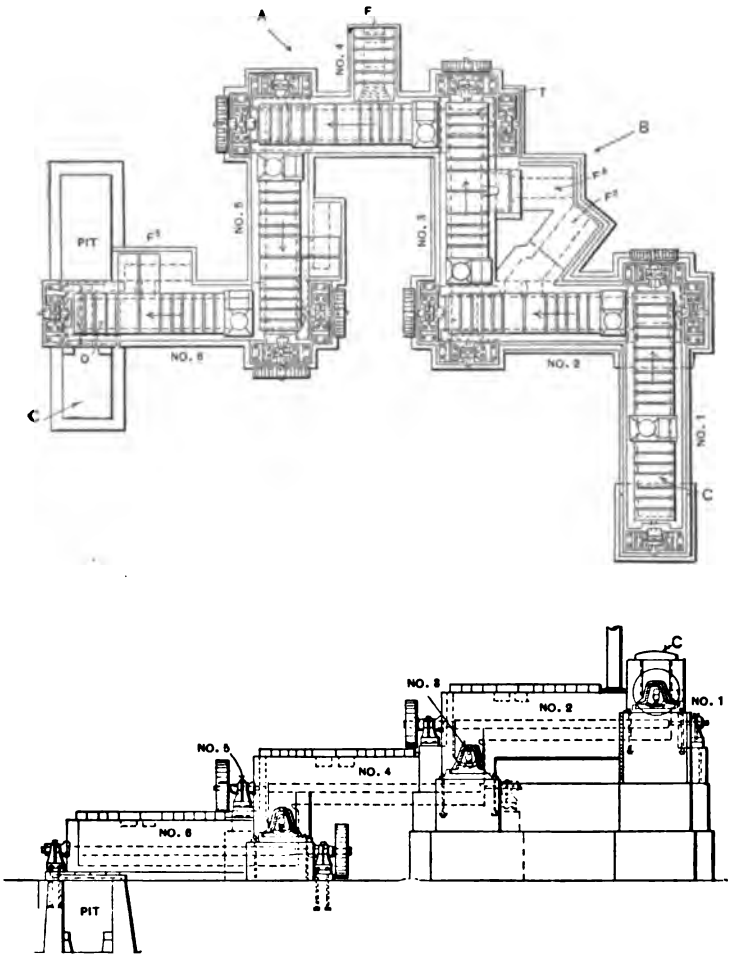

FIG. 2.—PLAN AND ELEVATION OF THE MIXERS.

passed to the same elevator which carries the dust to the mixers. Development has shown that it is not necessary to use the soft coal with anthracite-dust. However, this bin is used when experimental runs are made requiring the mixing of different materials with the dust.

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The Binder.—Coal-tar pitch, used as a binder, is received on the Plymouth Street side of the building. It is hoisted to the second floor by means of a barrel-hoist, where the staves are removed and the pitch is thrown into the binder melting-tank (the tank holding about 15 tons of pitch), and pumped by means of a rotary-pump into the storage or hot binder-tank, where it is kept heated.

The Mixers.—Fig. 2 shows a plan and an elevation, with the relative position of the connecting mixers and heat-flues, and Fig. 3 shows a section of a typical mixer unit. The number of units necessary in a mixer depends upon the material to be briquetted and the condition in which it is received. At this plant six were used, and have proved well adapted to the

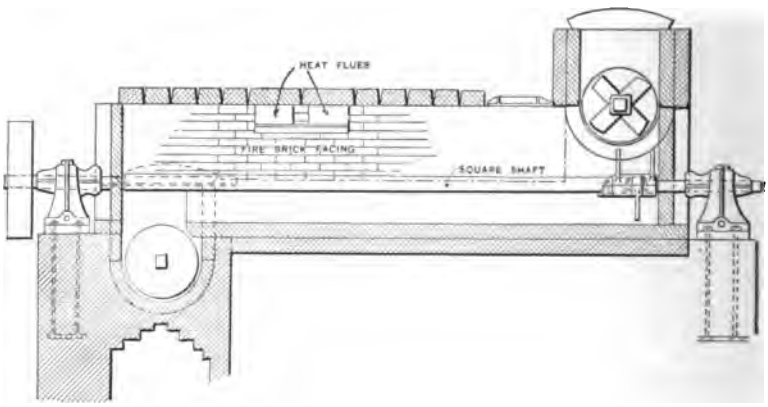

FIG. 3.—SECTION OF A TYPICAL MIXER.

handling of coal, hard and soft, wet or dry, coke-breeze, and even iron-concentrates.

The dust enters No. 1 mixer at *C*, Fig. 2, and is carried through mixers Nos. 1 to 6 in the direction indicated by the arrows to a point, *O*, mixer No. 6, and then by conveyor to the press. In passing through mixers Nos. 1 and 2 the dust is heated by the furnaces, F_2 and F_3, to drive off all the moisture. The coal-tar pitch, being previously heated, is pumped from the storage-tank by a small rotary-pump driven from a variable-speed countershaft, which regulates the percentage of pitch used.

The pitch is delivered to the mixer No. 3 at the point, *T*, Fig. 2, where it is atomized by means of a steam-jet.

The above apparatus and process are patented.

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The Press.—The roll type of press, shown in Fig. 4, is used, the rolls being built up of disks which are milled to form the pockets and are then assembled and bolted together on the shaft. This method, as well as the design of the briquettes, is patented. Two sizes of briquettes are made, $1\frac{7}{8}$ in. by $1\frac{7}{8}$ in. by $1\frac{1}{4}$ in. and $2\frac{1}{8}$ in. by $2\frac{1}{8}$ in. by $1\frac{3}{8}$ in. The briquettes are square "pillow" or "pin-cushion" shape. The smaller ones weigh 2 oz. and the larger 3.3 oz.

Cooling-Table.—The cooling-table consists of three endless belts composed of steel plates carried at their ends by sprocket-chains, the belts being placed one over the other and carrying

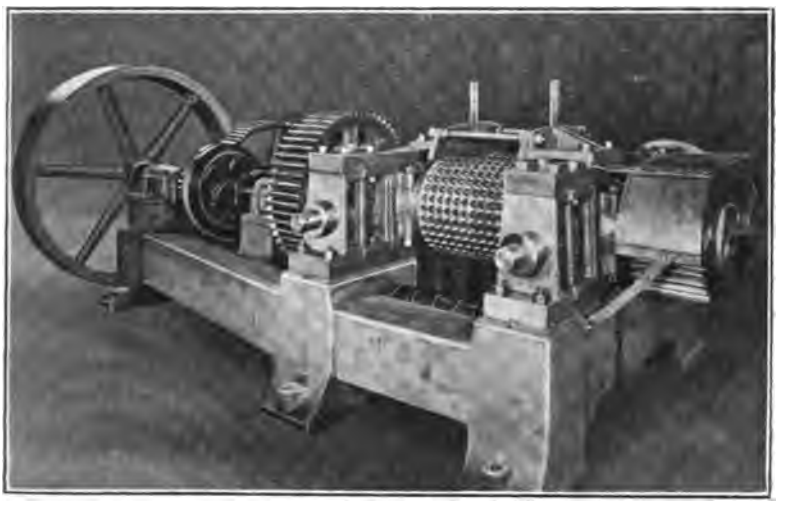

FIG. 4.—BRIQUETTING-PRESS OF THE NEW JERSEY BRIQUETTING CO.

the briquettes back and forth six times over a distance of 84 ft., making a total travel of 504 ft. The briquettes are then run into bins or loaded into wagons.

Fig. 5 is a view of the press built by the Zwoyer Fuel Co. for the Bankhead Mines, Ltd., Bankhead, Alberta, Canada.

Staten Island Plant.—The Briquette Coal Co., J. P. Egbert, Manager, No. 2 Stone Street, New York, N. Y., has just completed the construction of a briquetting-plant at Stapleton, on Staten Island. This plant is constructed for the purpose of using anthracite-dust with coal-tar pitch as the basis of the binding-material. The anthracite-dust is used as delivered. The plant does not possess any novelties in its design, except

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that there are two presses of radically different types. One of these is of German manufacture, built at the works of Schüchtermann & Kremer, of Dortmund. This press is of the plunger type, in which the manner of feed, compression and ejection is similar to the Johnson (English) machine used at the U. S. Geological Survey testing-plant at St. Louis, except that the disk containing the compressing-molds is set and revolves horizontally instead of vertically. The briquette is par-

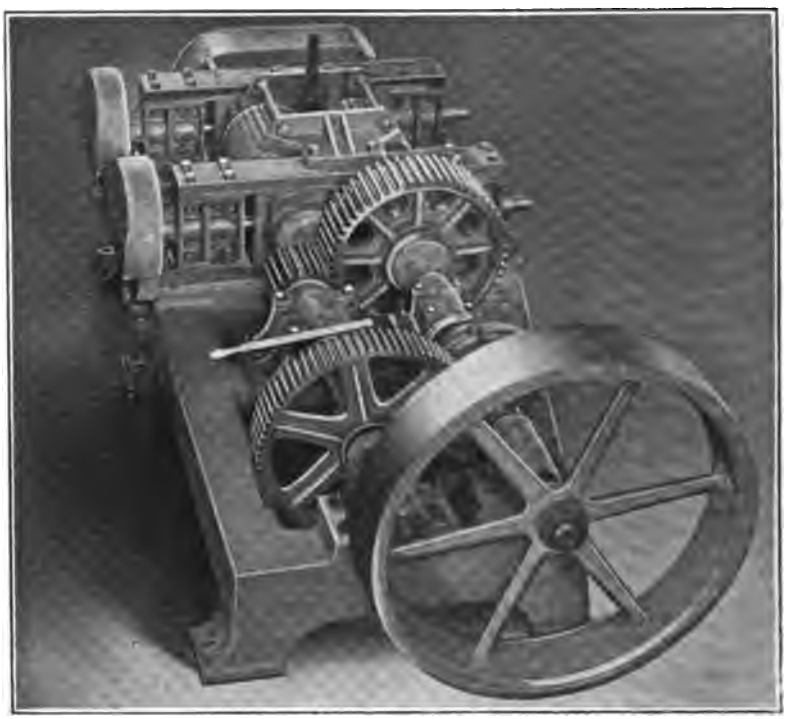

FIG. 5.—ZWOYER FUEL CO. PRESS, BUILT FOR BANKHEAD MINES, LTD., BANKHEAD, ALBERTA, CANADA.

allelopiped in shape with the end edges rounded. Its dimensions are 4.75 by 2.25 by 2.5 in. The briquettes weigh about 1.5 lb. each and have a specific gravity of about 1.24.

The second press is what is generally classed as the Belgian type, similar to the one described as the "American" machine used at the U. S. Geological Survey testing-plant. This particular machine was made at the works of H. Stevens, at Charleroi, Belgium. The product is of the eggette pattern, which

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is more desirable for domestic use than the larger briquette. The eggettes weigh about 5 oz., and have a specific gravity of 1.37. The manager of the company, Mr. Egbert, extended to me every courtesy possible, but unfortunately was not able to furnish any complete drawings or other illustrations, the plant having been constructed without them. The total capacity of this plant with both presses in operation is 120 tons of briquettes per day of 10 hr. The German machine will turn out 4.5 tons, and the Belgian machine 7.5 tons per hour.

South Brooklyn Plant.—Another plant, which has just been completed as this report is written, is that of the National Fuel Briquette Machinery Co., of New York, N. Y. This plant is located at the foot of Court and Smith Streets, Brooklyn, and close to the Gowanus canal, by which the materials to be used can be brought in barges and discharged at a minimum of expense. While intended to be operated upon a commercial basis, it may be considered rather as a demonstrating-plant. It is intended for the use of anthracite-dust with coal-tar pitch as a binder. The press is of the Belgian type, producing eggettes or "boulets," somewhat smaller than an ordinary hen's egg, and made exclusively for domestic use. The machinery used in this plant was patented in this country[4] by Hubert J. Debauche, Gilly, Belgium, and assigned to Mr. Robert Devillers, with whom I visited the plant, and to whom acknowledgments are made for courtesies extended. The eggettes produced by this plant are much smaller than those ordinarily made, weighing only about 1.5 oz. each, and having a specific gravity of 1.3.

North American Coal Briquette Co.—This company, with office at 177 Broadway, New York, N. Y., has been incorporated for the purpose of exploiting the Forst briquetting-process, which comprises chiefly the material to be used as a binder, part of which, though kept secret, consists principally of coal-tar pitch. The merit claimed for the secret ingredients of the binder is a great economy in the quantity of binder used for the manufacture of superior briquettes. The company has negotiated for the purchase of a Duprey (French) machine, and

[4] U. S. Patent No. 799,149, Sept. 12, 1905.

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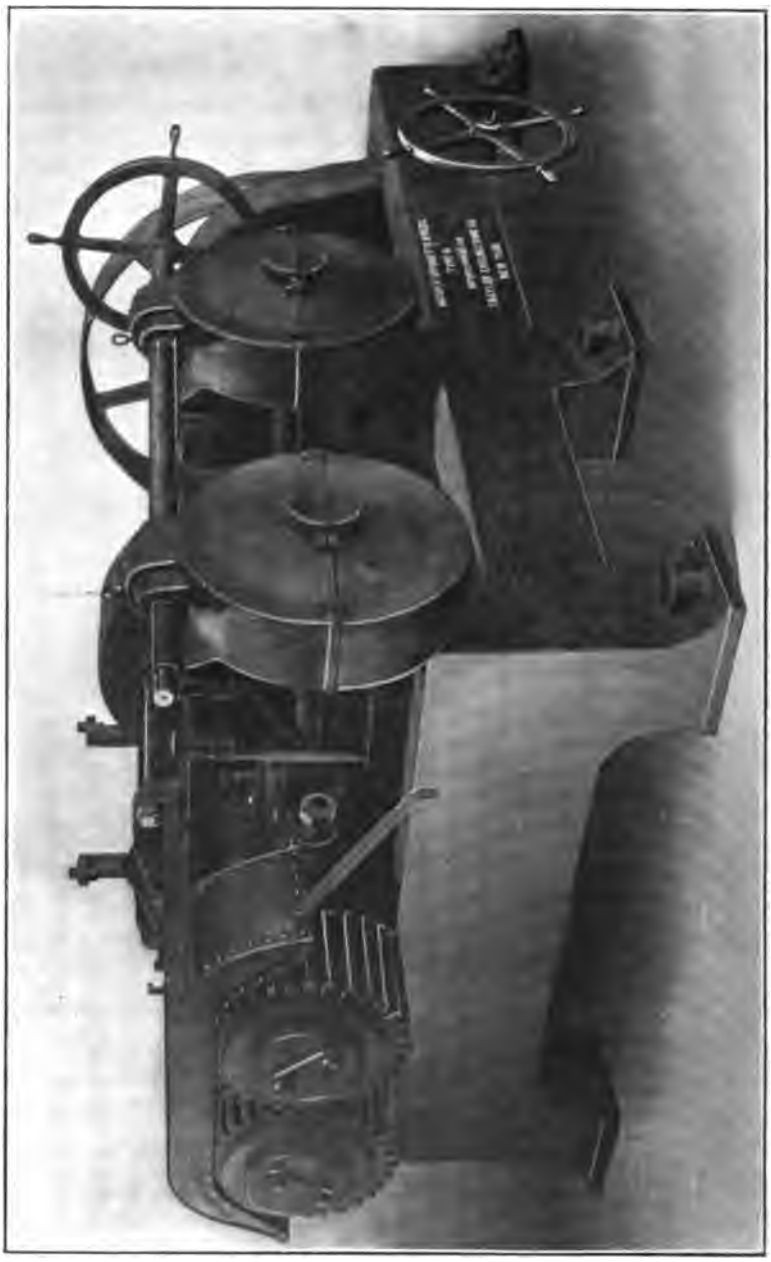

FIG. 6.—MASHEK BRIQUETTING-PRESS COMPLETE.

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has sent 10 tons of anthracite coal and one ton of binder to Paris for the purpose of demonstrating the claims made for the process controlled by the company.

The Mashek Briquetting-Process.—The briquetting-machine designed by Mr. G. J. Mashek (now with the Traylor Engineering Co., New York, N. Y.) has been described in detail by him.[5] The process was designed for the purpose of overcoming the objections to the use of briquetting-machinery which had developed, principally through the failure of certain foreign-made machines to meet the requirements of the American trade. When starting on the development of his plans, in 1903, the general type of machinery in use in Europe was that making the large rectangular briquettes, weighing from 7 to 20 lb. each, and these proved unsuitable to American use. In designing his press Mr. Mashek adopted the Belgian idea of molds contained in the peripheries of two tangential wheels, but instead of the eggette pattern, developed one which minimizes the blank spaces between the molds and produces a briquette of pillow or pin-cushion shape. Fig. 6 is a view of the Mashek press.

The Traylor Engineering Co. has recently built for Mr. E. B. Arnold a Mashek press, which has been installed at the foot of West 47th St., New York, N. Y. The building in which this is located was designed and erected for, and originally equipped with, a different type of machinery, but the briquettes made proved to be of a shape and character unsuited to the trade, and the cost of manufacture was also too high to enable the briquettes to compete with natural coal. When it was decided to substitute a Mashek press for the old one, it was also deemed advisable to use the same building, which is a substantial one, and also, as far as possible, the old machinery (such as elevators, shafting, power-plant, etc.), which was practically new and in good order, but which did not permit of the most desirable arrangement.

The new press installed has a capacity of about 14 tons per hr. of 2-oz. briquettes, but on account of using so much of the old equipment it is impossible to handle sufficient material to keep the machinery running at its full capacity, and it is now

[5] *Iron Age*, vol. lxxvii., pp. 1330 to 1333 (1906).

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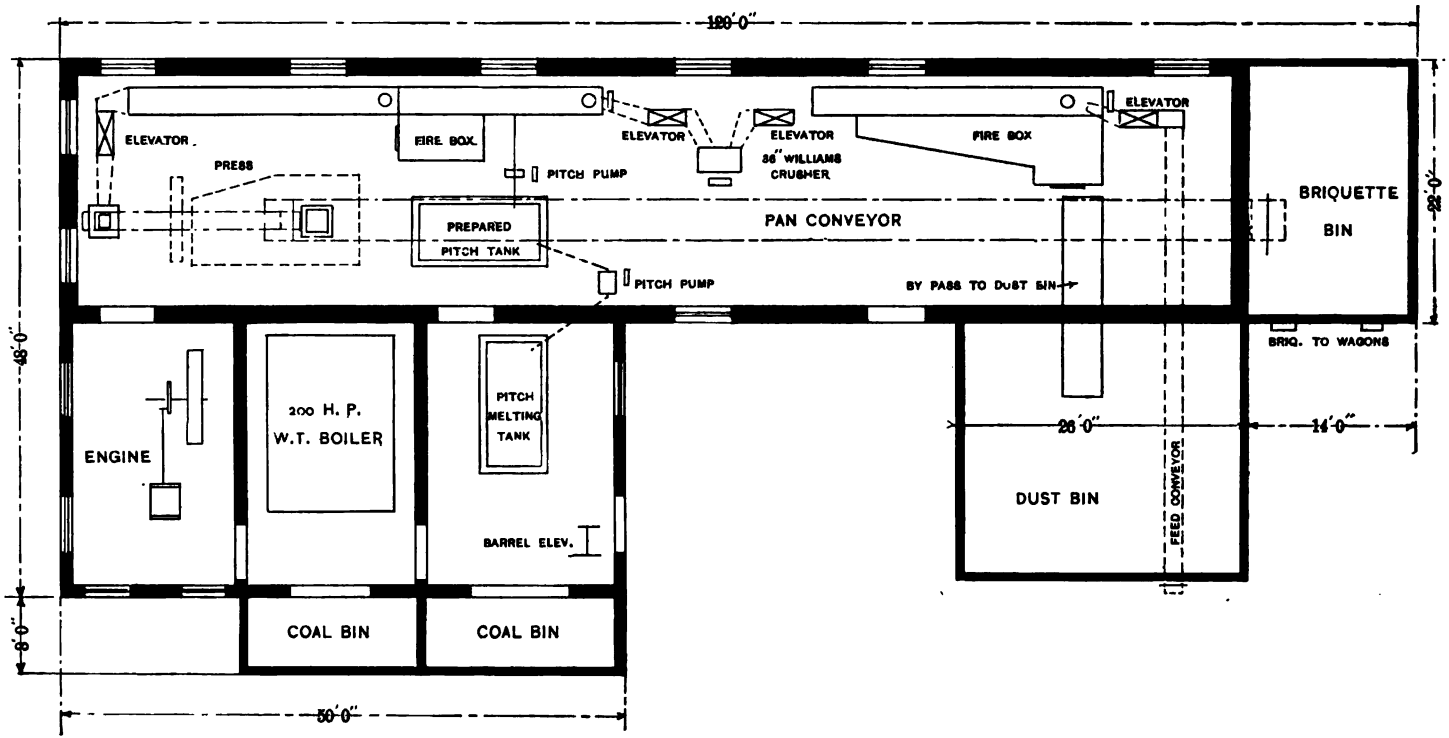

FIG. 7.—PLANT OF MASHEK PROCESS. (Plan.)

MASHEK PROCESS ANTHRACITE COAL BRIQUETTING.

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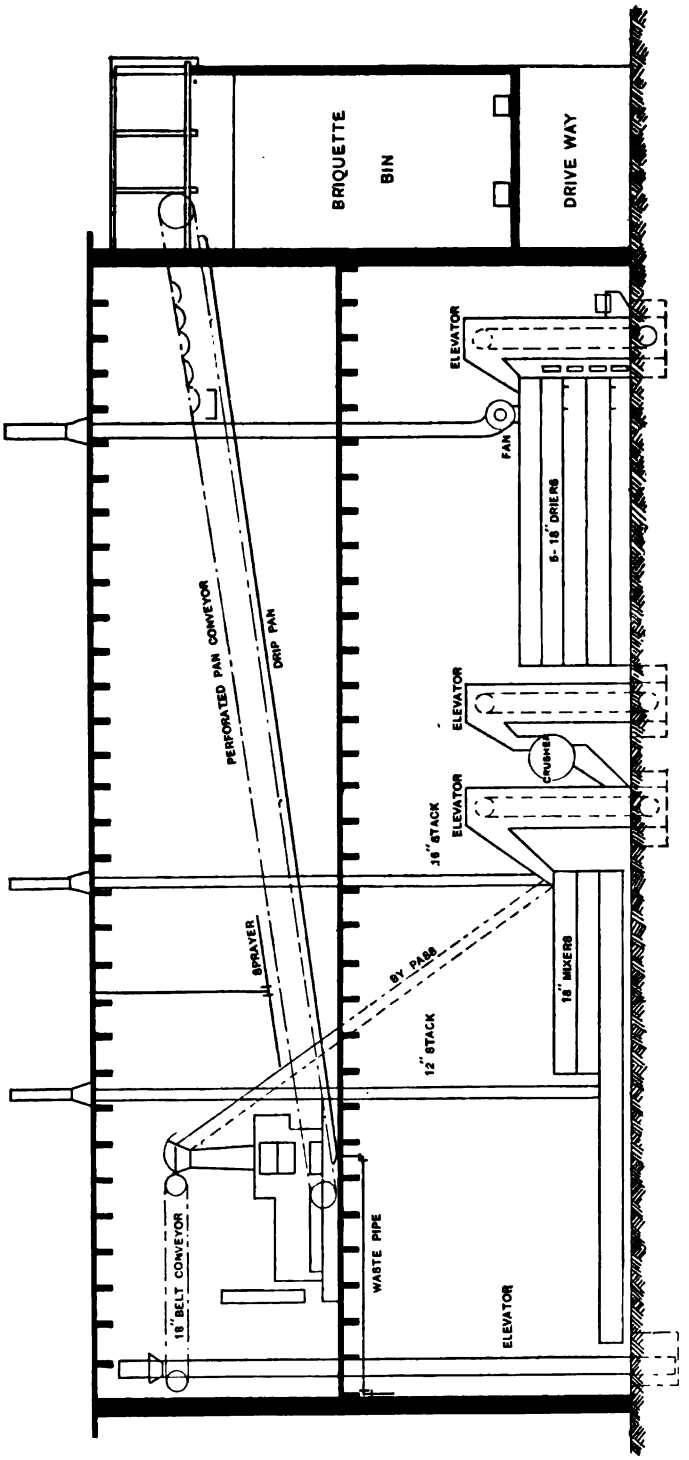

FIG. 8.—PLANT OF MASHEK PROCESS. (Elevation.)

PLANT USING MELTED COAL-TAR PITCH FOR BINDER.

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operated at the rate of about 10 tons per hr. The cost of labor, fixed charges and other expense being the same, the cost of production is slightly higher per ton of briquettes than would be the case if the plant were operated up to its maximum capacity. The size of the briquettes to be made has been determined by putting them on the market and selling them for domestic purposes, starting with 1-oz. briquettes and running up to 3-oz. It was found that the majority of users preferred a 2-oz. size, which corresponds with the stove-coal size of anthracite. The weight, of course, will vary with the nature of the dust from which the briquette is made, and it has been found that in using coke-breeze a 2.5-oz. briquette is most desirable, and about a 3-oz. if made of soft coal and lignite. The press is so designed that a change of the mold-shells can be made in about 2 hours.

The plan and elevation of the Arnold plant are given in Figs. 7 and 8. The arrangement of the machinery is shown in Fig. 8. The anthracite-dust is elevated to the dust-bin, from which it is drawn by a feed-conveyor so arranged that the feed is constant and can be regulated as desired. This conveyor discharges into a chain-elevator, which in turn discharges into a battery of five 18-in. rotary driers and heaters. These are super-imposed one above the other and all bricked in. The material is conveyed through these driers by means of screw-mixers until it passes into the following elevator.

On the side of these driers is constructed a furnace, the products of combustion from which are distributed into the driers through openings into the different units, so that no unit gets heat sufficient either to char the dust or to burn out the iron-work of the paddle-conveyor. An exhaust-fan draws off the products of combustion and the moisture. The temperature of the discharge-gases and moisture from the drier rarely exceeds 212° F. After the material passes out of the drier into the elevator it is elevated and dropped into a 36-in. Williams pulverizer, where the larger pieces are crushed, so that everything passes through about a 12-mesh screen. From the pulverizer the material is again elevated to another series of mixers and coolers similar in construction to the driers. The anthracite-dust at this point has a temperature of about

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300° F. The coal-tar pitch is here introduced by means of a pitch-pump so arranged as to deliver a definite quantity of pitch, as desired. Alongside of this last battery of mixers is a small furnace which heats the two upper mixers, maintaining an even temperature of the mixture and not allowing it to stiffen or set. From the last mixer the material drops to an elevator which takes it up to the second floor and discharges it on to an 18-in. belt-conveyor, which delivers the material over the press and into the hopper. The press is run continually, discharging the briquettes into a perforated-pan conveyor, which conveys them to the briquette-bin. The briquettes while in this conveyor are subjected to a heavy spray of water in order to cool and clean them.

The coal-tar pitch used in this plant is of the ordinary roofing-hardness; it is delivered by lighter on an adjacent dock and carted to the pitch-melting house, where it is melted in a tank 6 ft. wide, 12 ft. long, and 8 ft. deep.

This pitch-melting tank will hold about 22 tons of pitch, which requires in the neighborhood of 20 hr. to melt. After the pitch is melted and brought up to the proper temperature for use it is drawn off by means of a large pitch-pump into the "prepared-pitch tank," from which it is pumped into the mixers.

This plant requires about 125 h.p. to turn out 10 tons per hr. It has been in operation about two months and is said to be giving excellent results. The product is used almost entirely for domestic purposes, and commands the same price as the best grade of prepared anthracite coal in the New York market. A large portion of the output is put up in paper bags and handled by grocers and small coal dealers the same as charcoal or crushed coke. The bag trade caters to the poor people who do not buy in large quantities, and is a considerably cleaner method of distributing the product than that formerly used.

Out of the briquette-bin the briquettes are handled the same as ordinary coal, and experience has shown in this and other plants that abrasion or breakage averages about 3 per cent., which is slightly less than with ordinary prepared coal.

The cost of manufacture is as follows:

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Pitch:		
Using 6 per cent. of pitch at $10 per ton,		$0.60
Deducting increased weight of product due to 6 per cent. of pitch, and calculating product at $5 per ton,		0.30
Net cost of pitch,		$0.30
Fuel:		
For boiler, broken coal and screenings, broken briquettes, 4 tons per day of 10 hr., at $2.50 per ton,	$10.00	
Per ton of briquettes,		0.10
For heaters, driers and pitch-melting, 3 tons at $2.50 per ton = per ton of briquettes,		0.075
Labor:	*Per Day.*	
1 foreman,	$5.00	
2 pitch-melters,	3.50	
1 dust-bin man,	1.75	
1 engineer,	3.50	
1 man on second floor,	1.75	
1 man on ground floor,	1.75	
1 night watchman,	1.75	
1 oiler,	1.75	
	$20.75	
Per ton of briquettes,		0.21
Miscellaneous:		
Wear and tear, per ton of briquettes,		0.10
Lubricating oil, per ton of briquettes,		0.01
Insurance,		0.005
Interest on capital invested, $40,000, at 6 per cent.,		0.10
Office expense, telephone, stenographer and stationery, $2,000 per annum,		0.09
		$0.99
Anthracite-dust at $1.40 per long ton = per net ton of briquettes,		1.25
Total cost of briquetting,		$2.24
Re-briquetting 3 per cent. of breakage and abrasion, charging it back to plant as dust, per ton of briquettes,		0.06
Net cost per ton of briquettes,		$2.30
Wholesale selling-price in bin,		4.80
Net profit per short ton,		$2.50

PENNSYLVANIA.

It might be supposed that the briquetting-industry would have its greatest development in or near the anthracite-region of Pennsylvania, where a plentiful supply of raw material is available in the great culm-banks created through the many years of mining, and in the still large amount of fine coal pro-

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duced at the breakers for which no profitable market has yet been found. Up to the present time, however, there are but two briquetting-plants in operation in the State, one at Point Breeze, in the city of Philadelphia, and the other at Dickson, a few miles from Scranton. Both of these were put in operation in 1906. The plant at Dickson is in the immediate vicinity of the mine of that name, operated by the Delaware, Lackawanna & Western Railroad Co., and uses the fine coal or screenings, below marketable sizes, coming from the washery operated in connection with the mine. The owner of this plant, the Scranton Anthracite Briquette Co., withholds information relative to the details of its operations. I have been informed, however, by one of the officials of the company that the base of the binding-material used is coal-tar pitch, and that the plant is producing at the present time—April, 1907—from 300 to 325 long tons of briquettes per day. It is the intention to double this output by running the plant night and day. The briquettes are of the oval or eggette shape, the press being of the Belgian type, and similar to the "American" machine used at the testing-plant of the U. S. Geological Survey at St. Louis during the exposition period.[6] The entire product is taken by the Delaware, Lackawanna & Western Railroad Co. for use, principally, on its locomotives.

The other plant, at Point Breeze, is owned and operated by the United Gas Improvement Co., and was constructed for the purpose of utilizing the coke-breeze produced at the gas-houses of the company, and in this case, as in the case of the plant at Dickson, the product is not placed upon the market but is used by the company in its retorts for the manufacture of water-gas. I am indebted to Mr. W. H. Gartley, Engineer of Works of the United Gas Improvement Co., for the following detailed description of the plant, and for the accompanying illustrations, Figs. 9 and 10.

It has been found advantageous to use a mixture of anthracite-culm and coke-breeze, with from 5 to 7 per cent. of coal-tar pitch as a binder. The proportions of culm and coke used are variable, according to the quantity of material on hand. At the time I visited the plant—November, 1906—three parts

[6] *United States Geological Survey Bulletin* No. 261 (1905), and *Professional Paper* No. 48 (1906).

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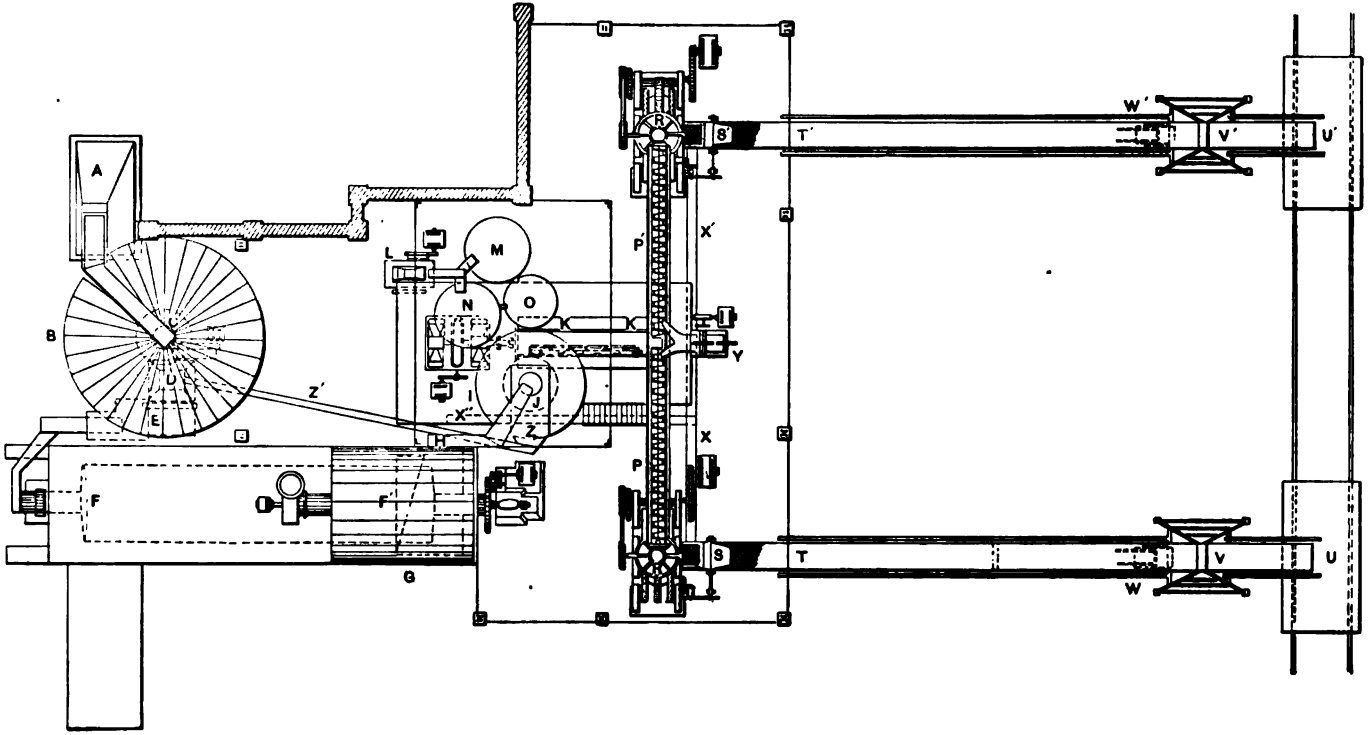

FIG. 9.—BRIQUETTE-PLANT OF THE UNITED GAS IMPROVEMENT CO. AT PHILADELPHIA, PA. (Plan.)

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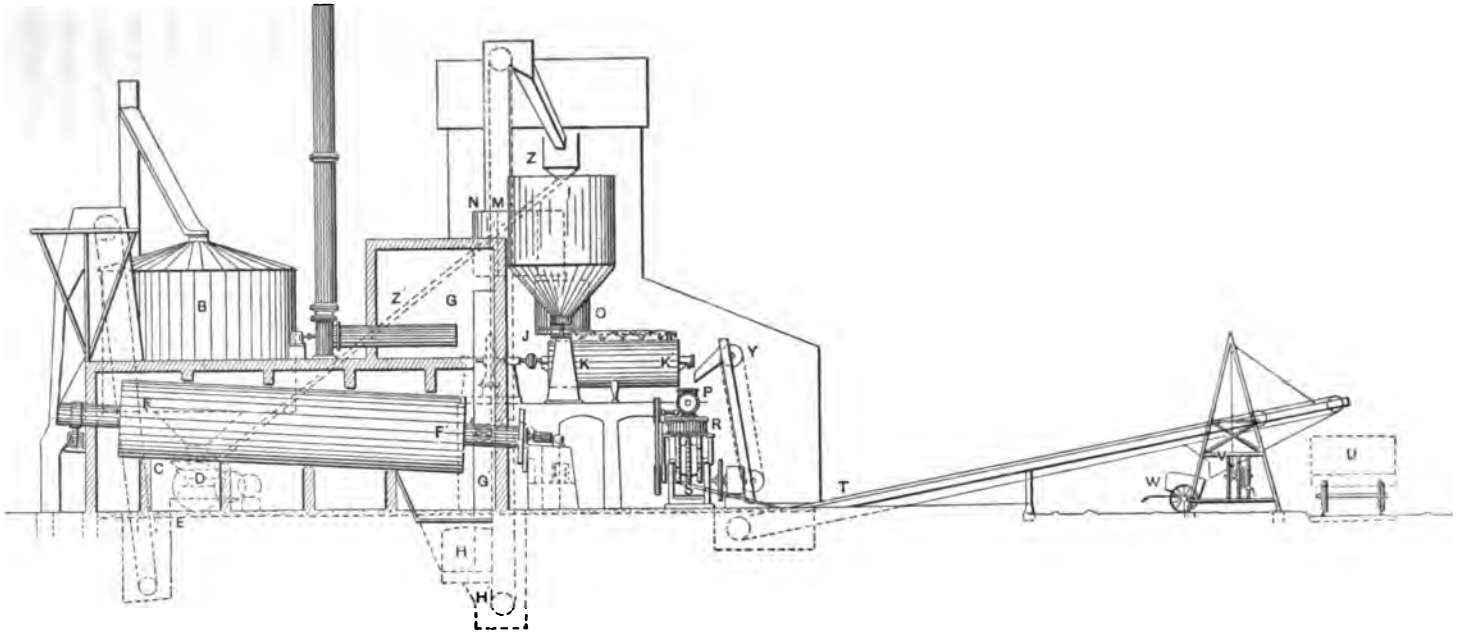

FIG. 10.—BRIQUETTE-PLANT OF THE UNITED GAS IMPROVEMENT CO., PHILADELPHIA, PA. (Elevation.)

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of culm to two parts of coke were being used. The press is of the Belgian type, producing eggettes about the size of a goose-egg. The rated capacity of the plant is 10 tons of eggettes per hr. It has been in operation regularly, producing 90 tons per 9-hr. day, except when it has been shut down for repairs and changes. Figs. 9 and 10 show the arrangement of the plant.

The breeze or screenings from the coke-screen (not shown on the diagram) of the coal-gas plant fall into a pocket or hopper, *A*. Into this pocket is also dumped the culm. The contents are raised by an elevator into the storage-tank, *B*, discharging through the funnel-shaped bottom on to the automatic feed-table, *C*, by which a measured stream of the material is continuously poured, part into the crusher, *D*, and part directly into the hopper, *E*, below the crusher. The material is then elevated and discharged into the drier, *F*, *F'*. The dried material, together with the dust from the dust-chamber, *G*, *G'*, of the drier is discharged into the hopper, *H*, *H'*, elevated and discharged through a shaking-screen into the storage-tank, *I*, located above the mixer. All material not fine enough to pass through the screen, *Z*, is returned by a spout, *Z'*, to the crusher, *D*. The dried material in tank, *I*, is discharged through the funnel-shaped bottom on to the automatic feed-table, *J*, by which a measured stream is continuously poured into the mixer, *K*, *K'*. Into the feed-end, *K*, of the mixer is also pouring a continuous stream of liquid-pitch through a positive measuring-faucet (not shown), driven from the driving-mechanism of the mixer through a variable-speed device (not shown). The pitch is brought into the building as broken from the pitch-bays of the tar-distillery, fed into the pitch-cracker, *L*, elevated and discharged into the large steam-heated pitch-storage tanks, *M*, *N*, where it is melted. From these tanks the melted pitch is drawn, as required, into the smaller steam-heated tank, *O*, to which the faucet, previously mentioned, is attached.

The warm, dry, and continuously measured crushed breeze and culm, together with the melted and continuously measured pitch, are thoroughly mixed and kneaded in the steam-jacketed mixer, *K*, *K'*. The mixed mass is discharged from the mixer, divided (device not shown) into two streams and carried by two mixing-conveyors, *P*, *P'*, allowing time for cooling and

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setting, into the feed-pans of the two presses, R, R', purchased in France. From the feed-pans the material is fed to the presses, which press out the eggettes and discharge them on to the shaking-screens, S, S', below, which screen them from the waste and fines. They are then discharged on to the woven-wire belt-conveyor, T, T', giving the eggettes time to cool and set, and conveyed either to the cars, U, U', or to the hoppers, V, V', from which the buggies, W, W', for the generator-house are filled.

The waste and fines from the shaking-screens, S, S', under the presses, are conveyed by conveyors, X, X', to the hopper, H', at the discharge of the drier.

Screenings from the eggettes taken from the storage-piles are returned by an elevator, Y, to the discharge of the mixer and assist in the cooling of the heated mixture.

California.

The manufacture of briquettes has shown more actual progress in California than in any other State of the Union. This has been brought about through the efforts to improve the fuel-quality of the rather low-grade California lignites, and has been encouraged by the high prices of the better grades of bituminous coal or anthracite brought into the State from Washington, the Rocky mountains, and eastern States, or imported from British Columbia, England, Australia, and Japan. It has also been encouraged by the cheap asphaltic pitch obtained from California petroleum, which not only serves excellently as a binder but adds to the calorific value of the briquetted fuel.

The first plant to be put into successful operation in California was one built at Stockton by the San Francisco & San Joaquin Coal Co. The plant was completed in 1901, and when running at full capacity could produce 125 tons of briquettes per day. The fuel used was lignite from the Tesla mines, in Alameda county. The plant was, unfortunately, entirely destroyed by fire in 1905 and has not been rebuilt. It is stated that the plans of the company were to rebuild the plant at San Francisco, but these were upset by the earthquake and fire which destroyed a large part of that city in April, 1906. A complete description of the Stockton plant by the designer of

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the presses, Mr. Robert Schorr, of San Francisco, has been published.[7] The briquettes produced at this plant were round, convex lenses or "boulets," which weighed from 6 to 8 ounces.

The Western Fuel Co., of Oakland, completed, early in 1905, a briquetting-plant also designed by Mr. Schorr.[8] In mechanical construction this plant differs materially from the one destroyed by fire at Stockton. The shape of the briquettes is cubical instead of "boulet." The advantage claimed for the cubical shape is that the briquettes ignite more readily, though it is admitted that in handling the same mechanically there is more waste.

The capacity of this plant is 480 briquettes per min., or 8.5 tons per hr. The fuel used is coal-yard screenings from lignites, anthracite, and sub-bituminous coals, with about 7.5 per cent. of asphaltic pitch. This pitch is obtained by the distillation of California crude petroleum. The temperature of the still, for the production of the proper grade of pitch, is about 600° F. Some difficulty has been experienced in securing the right quality of pitch on account of the tendency of the refineries to "rush the stills," their aim being the securing of refined oils rather than pitch. An excellent grade of asphaltic pitch is obtained by keeping the stills at a temperature of 500° F., using a vacuum to force the distillation. Grade "D," the quality best adapted for the purpose, is fairly hard up to 60° F., but begins to soften above that point. It becomes liquid at 250° F., and has a specific gravity of from 1.05 to 1.1.

Before the earthquake, the Western Fuel Co. paid $10.50 per ton for the ordinary pitch "D" delivered at its plant, while a properly and carefully prepared pitch was worth from $12 to $13.

Owing to the enormous building-activity in San Francisco since the earthquake, the demand for asphaltum for roofing-materials has increased in leaps and bounds. Consequently, there is a great scarcity, and the price per ton now ranges from $14 to $20. This scarcity necessitated many shut-downs of the plant at Oakland, and for that reason the company is negotiating for the importation of coal-tar pitch from the East and from Europe. As three new refineries are contemplated, conditions may gradually return to their normal stage.

[7] *The Engineering and Mining Journal*, vol. lxxviii., p. 262 (1904).

[8] *The Engineering and Mining Journal*, vol. lxxx., p. 389 (1905).

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All of the coal purchased and used by the Western Fuel Co. is brought in ships and is unloaded by electric hoists into receiving-bins. When drawn from the storage-bins it is screened, all material passing through the perforations dropping into auxiliary bins, from which it is fed into a Williams crusher. The disintegrated coal from the Williams crusher is elevated into the iron hopper of an automatic feeder which feeds into the coal-heater. The heated coal enters the mixer and meets there the binder. The mixer as well as the binder-distribution and the tempering of the mixture embody some novel features.

The prepared material is conveyed into the feed-hopper of a Schorr press, style "A," which is belted for 6 rev. per min. At that speed 480 briquettes of 9.5 oz. weight are discharged in one min., or more than 17,000 lb. per hr. The briquettes are of a rectangular shape, 2.75 by 2.5 by $1\frac{7}{8}$ in. thick, with rounded corners and branded with a "W." They are uniform in size, and have a specific gravity of about 1.22.

All wearing-parts of the press are lined with phosphor-bronze, and they are thoroughly lubricated under an air-pressure of 40 lb. per sq. in. Oil is also atomized and sprayed into the molds and upon the plungers.

The briquettes drop upon a short conveyor delivering the same to another one located outside the building. At this point provision is made to sack the briquettes for the local market, or to take them up to the top of the storage-bunkers, where arrangements are made to discharge the briquettes into cars or to distribute them into the bunker-compartments. The average output is 64 long tons per shift of 8 hr., four men being employed, one of them getting $4, one $2, one $3, and one $2.75 per day, which makes about $0.20 per ton of briquettes. By running 24 hr. more than 200 tons could be made, which would reduce the labor item to about 14.5 cents per ton. This can be further cut down by speeding up the press to 7 rev. per min., which would produce 560 per min., or 20,000 lb. of 9.5-oz. briquettes per hr. With a forced-feed attachment a further increase in speed may be possible.

Since the foregoing was written, wages have been increased considerably, most of the men getting $3.50 per shift, working through the lunch-hour.

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The present pressure-arrangement was tested up to 48,000 lb., exerted upon two 2.5- by 2.75-in. surfaces, making more than 3,400 lb. per sq. in. The adjustment is placed to give about 2,900 lb., which is ample, and makes a better-burning briquette than when a greater pressure is used. The press is designed for a maximum pressure of 6,000 pounds.

The briquetting-press has been described by Mr. Schorr substantially as follows:[9]

"In this press two sole-plates with heavy bearings are arranged to carry a stationary steel shaft, upon which a large spur-wheel is revolving, driven by means of gearing, countershaft and friction-clutch pulley. The spur-wheel rim is made integral with a mold-ring which has a series of holes and sliding plungers (pistons) therein. The pistons are under continuous control of cams which are supported by heavy shields. The pistons are released from the camway only when the final pressure is applied, and this is done by a large wheel with steel tire, pivoted in two levers.

"This wheel is pressed against the piston-heads by means of an adjustable spring, which permits a perfect regulation of pressure up to 4,000 lb. per sq. in. After leaving the pressure-wheel, *i.e.*, after the briquette is made, the plungers are gradually forced forward to eject the briquettes, which drop upon a vibrating discharge-chute.

"The pistons are then gradually withdrawn, and in passing the feed-box the cavities become filled with the mixture of coal and pitch. At the end of this feed-box all surplus material is scraped off by a steel plate. After passing the scraper-plate the pistons are gradually forced in, pressing the material against the resistance-block, which is supported by the main shaft. This pressure is effected by a cast-iron stand with phosphor-bronze liner.

"When the pistons are about 0.5 in. from their terminal, they strike against the rocking pressing-wheel and are forced home. In this way the briquettes are made, and the play repeats itself with every revolution.

"The machine is entirely self-contained, and it is claimed that there is no possibility of its getting wrecked by overfeed or obstruction. It is also claimed that as the pressure is applied slowly and gradually, this type of press permits briquetting mixtures containing from 13 to 14 per cent. of moisture, and that this is an advantage not possessed by intermittently-acting presses. Up to the present time two press designs have been made, the one with two rows of 2-in. cylindrical molds, and the other with two rows of 2.5- by 2.75-in. rectangular shapes with rounded corners. There is no difficulty in making other shapes and heavier briquettes. A simple arrangement permits working with half the capacity whenever desired. No complications are presented if it is desired to have more than two rows of molds, and the type of press can be built for a much larger capacity. On the other hand, should the market for briquettes be limited for some months in the year, the capacity can be cut down without requiring any change in speed or other alterations.

"From 80 to 120 briquettes are made for each revolution, the number depending on the size and shape of the briquettes, which govern also the capacity. The

[9] *Engineering and Mining Journal*, vol. lxxx., p. 627 (1905).

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same varies from 6 to 24.5 tons per hr. The briquettes are uniform in size and plainly branded with 'W.' "

Mr. Schorr says that all wearing-parts of the machine can be quickly and cheaply replaced. The lubricating is done by an air-compressor and oil-atomizer.

The press is especially adapted for the manufacture of small briquettes, and the use of such in preference to large blocks is obvious. Small briquettes can be readily shoveled into furnaces, while the large ones have first to be broken up, thus causing labor, waste, and dust.

A briquetting-plant of an entirely different type, designed by Mr. Chas. R. Allen, was built and put in operation by him during 1905 at Pittsburg, at the junction of the San Joaquin and the Sacramento rivers, about 50 miles from San Francisco. This plant, as originally projected, was intended to utilize the lignite produced by the Pittsburg Coal Mining Co. at Somersville, but the enormous increase in the production of oil in California has had such a demoralizing effect upon the coal-trade generally that there has been little or no market for lignite during the last two years, and these mines have been shut down. The material used has, therefore, been screenings obtained from the coal-yards of San Francisco, the binder (here, as at other plants in the State) being asphaltic pitch. The screenings are sold at less than the cost of mining lignite, and as long as the supply of this material is available at such prices it will continue to be used.

The methods of preparing the briquetting-mixture differ somewhat from those used at other plants in that the binder is passed, together with the fuel, through the retorts under a high degree of heat. This, it is claimed, insures an intimate and thorough mixture, each particle of fuel being impregnated with the binder. This treatment, it is asserted, prevents the binder from being consumed before the coal is ignited, which is apt to be the case, particularly with lignites, if the mixing is merely superficial.

Mr. Allen claims that in his process the nature of the fuel is changed so that the lignite partakes of the character of bituminous coal, the briquettes remaining firm and hard until entirely consumed. He claims also that the process possesses as much of novelty and value as the press.

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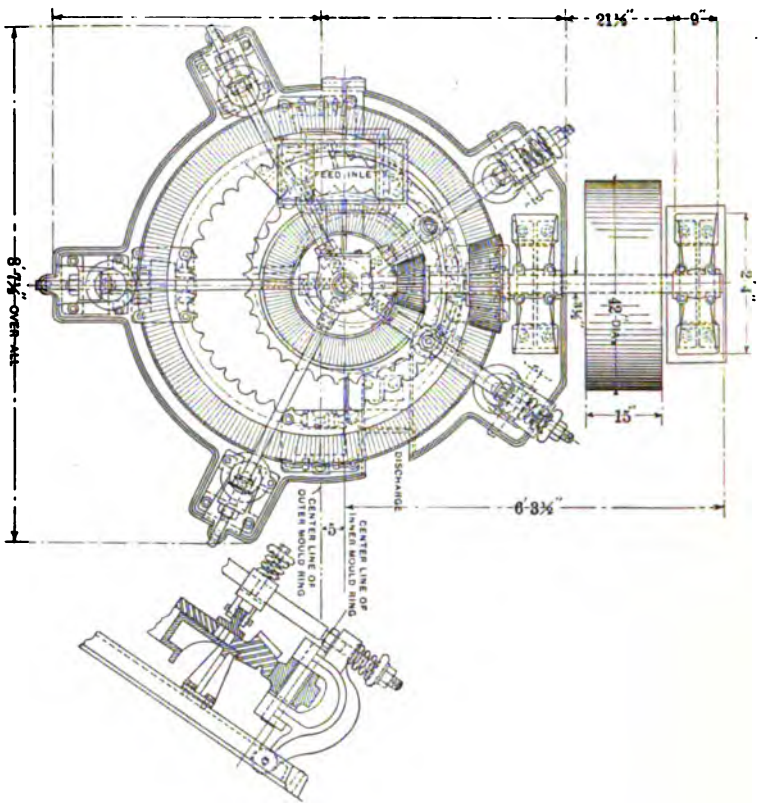

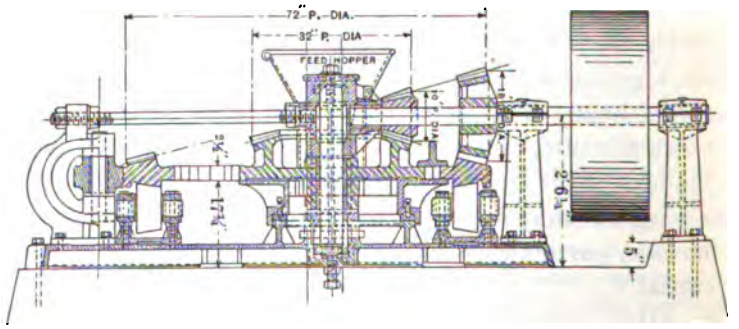

FIG. 11.—ALLEN BRIQUETTING-MACHINE. (Plan and Elevation.)

As shown in Figs. 11 and 12, the compressing-machine consists of two non-concentric rings horizontally placed, one within the other, the periphery of the smaller one being cor-

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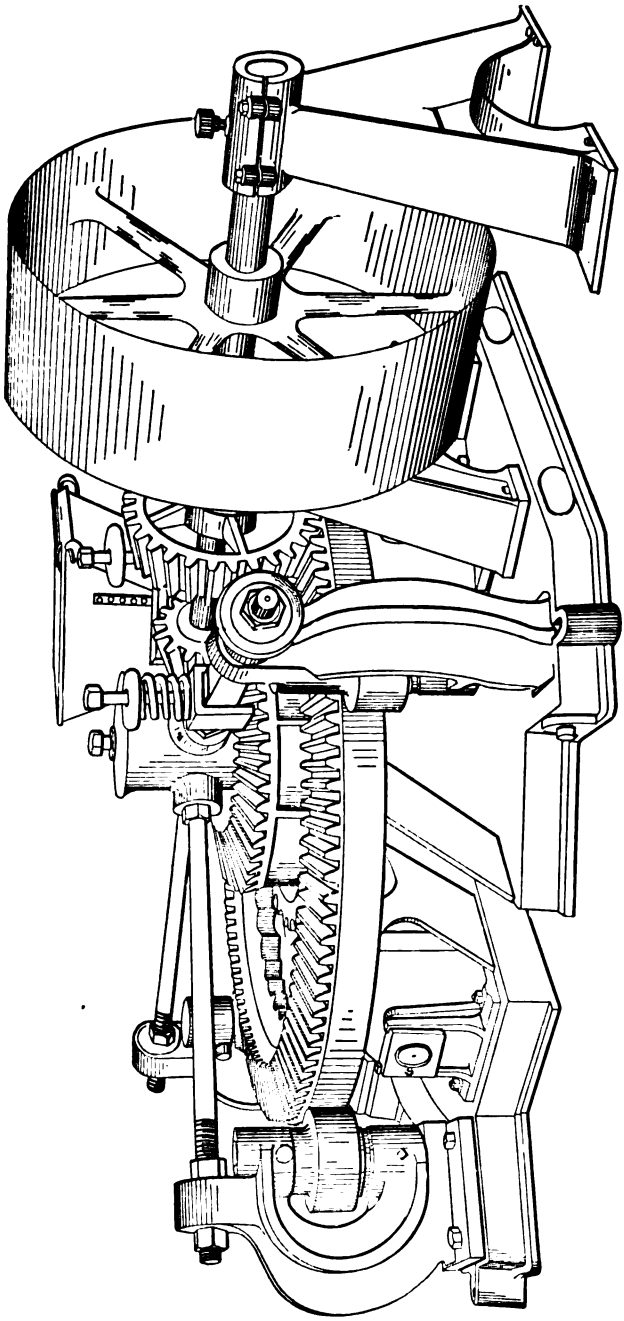

FIG. 12.—ALLEN BRIQUETTING-PRESS. (Perspective View.)

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rugated, or scalloped, and engaging with similar corrugations in the inside of the larger ring. The briquetting-mixture is fed into a hopper one-fourth of a revolution of the smaller ring from the point of compression, and the amount of pressure is regulated by the distance of the feed from the point of compression; that is to say, the hopper may be placed further away if a greater pressure is desired, or nearer, if the pressure is to be reduced. Relief from an excess of pressure is provided for by two heavy spiral springs on the outer bearings, and two over the upper pressure-plate, the lower pressure-plate being fixed. The relief-springs are shown in Fig. 11. The machine has been operated without using any of the springs, with the result that when there was a surplus of feed the operating-belt was thrown off through the choking of the machine.

The Allen machine is patented.[10] The briquettes, as now made, are approximately cylindrical in shape, with flat ends. They weigh from 8 to 10 oz. each, and have a specific gravity of 1.14. It is Mr. Allen's intention to reduce the size of the briquette and change its shape by having the smaller ring of the press made without corrugations. This will be done in order to meet the demand for a briquette better adapted for domestic use.

The plant is at present turning out about 5 tons of briquettes per hr., at a moderate running-speed. With a smaller briquette the production per hour would be decreased, with the same speed, while with an increased speed the same production could be maintained, even with the smaller briquette.

The Standard Coal Briquetting Co., of Oakland, constructed in 1905 a plant designed by a Mr. Crawford. An accident to the press shortly after being put in operation practically wrecked it, and the enterprise was unsuccessful.

Another plant beginning operations in 1905 was a small plunger type of press designed by Mr. A. Demetrak, and built by the American Briquetting Co. (afterwards reorganized as the Ajax Briquetting Co.), of San Francisco. It was destroyed by the earthquake and fire of April, 1906, and has not been rebuilt. The plant had a capacity of about 15 tons a day, using Coos Bay, Ore., lignite, sometimes mixed with coal-yard screenings, and asphaltic pitch.

[10] U. S. Patent No. 851,007, April 23, 1907.

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The United States Briquette Co., of Stege, Contra Costa county, has undertaken the manufacture of briquettes from a mixture of peat and California crude petroleum. This plant had not been completed at the time of writing this report, but some briquettes made of the mixture in an experimental way are interesting productions. They give promise of a method of using California oil as a domestic fuel, the peat, on account of its spongy character, acting as a carrying-vehicle for the oil, and at the same time performing duty as fuel. The briquettes are cubical in shape and of attractive appearance. They weigh about 10 oz. each and have a specific gravity of 1.3.

It is claimed that they are as well adapted for steam-raising as for domestic purposes, giving an intense heat under forced draft, and burning freely under ordinary draft; that they can be handled without waste from breakage, and that they leave a minimum of ash and do not clinker.

Arizona.

The Arizona Copper Co., Ltd., of Clifton, installed, during 1905, a briquetting-plant purchased from Messrs. Yeadon, Son & Co., of Leeds, England. The plant was put in operation in September, 1905, and, during the first six months of 1906, produced 690 short tons of briquettes of a total value of $4,830, or an average of $7 per ton. About 300 tons were produced in the experimental runs made in 1905. It was installed for the three-fold purpose of utilizing coke-breeze, which is without value and non-usable as such; for securing better efficiency out of the slack coal (Gallup) which is used as fuel; and for obtaining a fuel that could be stored without material deterioration, and without danger of spontaneous ignition. Mr. James Colquhoun, president of the company, affirms that what economic advantages are secured are from the first and third operations. In the briquetting of the coke-fines or breeze a profit of about $4 per ton is made in the conversion of a formerly-wasted material into a usable fuel. In using the Gallup, N. M., slack, which is of sub-bituminous, or black lignite, quality of coal, the expense of briquetting brings the total cost up to approximately $6.80 per ton, or about the same as that of the lump coal obtained from the same source, although the price for the slack at the mines is very low when

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compared with that of lump coal. The briquettes have been found to burn freely and satisfactorily under locomotive and stationary boilers, and appear to be equal to the best of Gallup lump coal, but no laboratory-tests of calorific power have been made.

The real profit in the briquetting of this coal is in the superiority of the briquettes over lump coal for stocking-purposes. They stand weathering perfectly, while the lump coal disintegrates upon exposure, loses a portion of its combustible gases, and becomes in time a very inferior fuel. It is also liable to spontaneous combustion, while the briquettes are not.

In the making of the briquettes 92 per cent. of the coal is mixed with 8 per cent. of California asphaltic pitch. The capacity of the plant is 2.5 tons of briquettes per hour.

The following description of the process at Clifton has been furnished by the company. In design the press is similar to the one used by the U. S. Geological Survey coal-testing plant at St. Louis. This was designated as the "English" machine, and has been described in the reports of those tests.[11]

The process of making briquettes of coal- or coke-fines is that of Yeadon, Son & Co., of Leeds, England. The fines are fed from the bins into the boot of a bucket-elevator, which discharges them into the hopper at one end of a mixer, where they are mixed with pitch that has previously been broken in a pitch-breaker into pieces of 0.5 in. maximum size. The quantity of pitch found to give the best results is about 8 per cent.

From the mixer the material is sent into a disintegrator, which thoroughly pulverizes the coal and pitch into grains of 2-mm. size and under. It is then elevated and passed into a heater, where it is subjected to the action of live steam, which gives the pitch sufficient fluidity to bind the other ingredients.

From the heater the material drops into a pug-mill which, while stirring the mass, sweeps it into a false bottom. This false bottom is behind the disk of the briquetting-machine, and, at each revolution of the main shaft, the material is rammed into a pair of compartments in the disk. The disk contains eight pairs of such compartments, and at the same time that a pair of briquettes is being rammed into the disk on one side,

[11] *United States Geological Survey Bulletin* No. 261 (1905), and *Professional Paper* No. 48 (1906).

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nother pair is being compressed on the opposite side, while third pair is being pushed out from the top of the disk nto an endless-belt conveyor, which delivers the briquettes to he side of a railroad-car in front of the building. The briuette-disk is made to revolve intermittently in eight periods o each complete revolution. During the pause in each period, he three operations referred to take place simultaneously.

The capacity of the plant is 25 tons per 10 hr. It is arranged to mix three ingredients into material for briquettes, ut at present only the fines from coal or coke and pitch are sed. The briquettes are rectangular in shape and weigh pproximately 4 lb. each.

Michigan.

The Semet-Solvay Co., of Syracuse, N. Y., has recently completed the construction of a briquetting-plant at Del Ray, to be perated in connection with the by-product coking-ovens and hemical-works installed there several years ago by the same ompany. The installation of the briquetting-plant was begun bout two and a half years ago. As originally constructed, the riquetting-machine was a reciprocating-press of English make, ut after carefully working out the process the company came o the conclusion that a press of the reciprocating type is dapted only to large briquettes, whereas the markets for which his product was intended demanded a small briquette for the omestic trade.

As the result of the experience gained with the English nachine, the company has developed a process for the manuacture of small briquettes, and although this plant is just beinning to run it gives excellent promise.

The process consists essentially of the intimate mixing of inely-powdered pitch of proper quality and consistency with ulverized coal, so that theoretically each particle of coal is oated with the fine pitch. The mixture is then brought up to he proper temperature with steam, or steam and hot water, and s fed to a rotary Mashek press built by the Traylor Engineerng Co., of New York. The output of the plant is from 10 to 15 ons of briquettes per hr. These are from 2.5 to 3 oz. in weight, nd about $1\frac{1}{8}$ in. square, shaped somewhat like a miniature sofaillow. This shape is satisfactory for shoveling and for handing in household stoves and furnaces. The company is using

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a portion of coke-breeze with the coal and pitch, with a view to making use of the breeze from its coke-plant, and it is also experimenting on the best mixtures and the best grades of coal. The briquettes made so far are said to burn well and to give no smoke, except a slight puff when they are first thrown on the fire. As the plant is not yet in full running, some minor adjustments are still being made to perfect the product, but the operators are much encouraged by results so far obtained, and expect within a short time to be making a thoroughly satisfactory and commercial product.

Missouri.

Renfrow Briquette Machine Co.—During the summer of 1903, Governor W. C. Renfrow, of Oklahoma, became financially interested in a briquetting company in St. Louis. In the fall of the same year Mr. E. D. Mizner, of Hamilton, Ont., visited St. Louis to make a report for some Canadian interests relative to the purchase of the Canadian rights for the patents of this company. The results of these investigations, and the efforts of Governor Renfrow to force the briquette company to deliver a machine, ended in the bankruptcy of the company. In October, 1903, an agreement between Governor Renfrow and Mr. Mizner was made by which Mr. Mizner was to build a briquette-machine which would overcome the difficulties encountered with the former press. No company was organized at that time, but contracts were drawn satisfactory to the people interested.

The following spring, Mr. Mizner built the first Renfrow press, which made briquettes 2 in. in diameter, weighing about 4 oz. This press had some of the essential features of the present Renfrow press but made briquettes only at one end of the stroke; that is, 12 briquettes per revolution. After this machine was built it was discovered that the briquettes were too small and that the construction of the machine was too light. Mr. Mizner also developed the idea of making briquettes at both ends of the stroke, thus doubling the capacity of the machine. It was decided to build a much heavier machine, and one making a briquette 3 in. in diameter. Changes were also made in the method of mixing and heating the material. The cast-iron vertical heaters of the original press were supplanted

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with horizontal jacketed heaters, using ordinary spiral-conveyor flights for mixing and handling the fuel. To this was added a short vertical heater acting as a reservoir, into which live steam was admitted just before the mixture was delivered to the molds.

The briquetting portion of the Geological Survey coal-testing plant at St. Louis during the Exposition has already been described.[12] After the close of the Exposition the "American" machine, installed by the National Compressed Fuel Co., of Chicago, was removed, and early in 1906 the remaining portion of the briquetting-plant was destroyed by fire. In rebuilding the plant provision was made for the installation of a Renfrow briquette-machine.

This machine was completed in the fall of 1905. Mr. C. T. Malcolmson, of the U. S. Geological Survey testing-plant, inspected this press at the shops of the Ramming Machine Co., at St. Louis, and burned some of the briquettes under a boiler at that plant. Difficulties were developed from the fact that the fuel remained too long in the vertical heaters, and some trouble was also experienced in getting the material from the die-filler to the die proper. Occasionally briquettes would stick in the dies, resulting in a double charge, which finally crippled the machine. Provisions were then made to overcome these difficulties and the machine was rebuilt. The new machine was first tested in March, 1906. The heating-capacity was increased so that the charge remained in the conveyors about 15 min. before reaching the dies, thus allowing the material to become thoroughly heated and the melted pitch to mix with the coal. Brushes were added to insure the charge being carried to its proper position in front of the die, and an ejector, operated by a magnet, insured the delivery of the briquettes from the ends of the plungers. Many of the parts of the machine were strengthened, and steel and bronze substituted for cast-iron in the wearing-parts. The results of the tests on this machine made under the supervision of Mr. Malcolmson for Mr. J. A. Holmes, Expert in Charge of the St. Louis fuel-testing plant, at the testing-plant of the company, resulted in a contract for the rental of this machine by the

[12] *U. S. Geological Survey Bulletins* Nos. 261 and 290 (1905), and *Professional Paper* No. 48 (1906).

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Government. In May, 1906, the first successfully-operating Renfrow machine was installed at the fuel-testing plant. It is shown in Fig. 13.

FIG. 13.—RENFROW BRIQUETTING-PRESS AT THE GEOLOGICAL SURVEY TESTING-PLANT.

The result of the tests which have been made on the Renfrow machine from May, 1906, to March, 1907, indicated that the design of this press was, in the main, satisfactory, and that

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the difficulties experienced were due almost entirely to bad or weak construction of the machine. This machine was the result of many changes, while in many cases it was impossible to strengthen the weak parts owing to the limited space, or the fact that the size of the part was fixed by the original design.

The difficulties encountered in operating this machine at the coal-testing plant soon indicated its weaknesses, and as a result the Renfrow Co. designed and built two new presses, one of which was installed and is now being operated by the Western Coalette Fuel Co. at Kansas City. The other is now ready for delivery to the fuel-testing plant at the Jamestown Exposition. The new press makes a briquette 3.25 in. in diameter, weighing about a pound. The machine which was operated at the St. Louis fuel-testing plant could not be depended upon to deliver more than 1,000-lb. pressure per sq. in. on the briquettes without seriously straining the frame of the press. The new machine will deliver a maximum pressure of about 2,500 lb. per sq. in. without straining the machine. All of the wearing-parts not under pressure are made of bronze so as never to become corroded, while the dies, made of case-hardened steel, are kept clean and bright by the abrasive action of the fuel. The cams and rollers, originally made of chilled cast-iron, are made of case-hardened tool-steel in the new machine, and the design of the housing has been so changed that any of the parts can be removed without dismantling the machine. Provision has also been made in the new machine so to feed the heaters that they will always run clean and at the same time keep a full load in the chamber above the die-filler. This chamber, while closed in the old machine, is open in the new, which allows the operator to regulate the supply of fuel to the press at all times. The plungers are arranged to make it practically impossible for a double charge to enter the press, and the length of the spring behind these plungers has been increased so that a double charge would not affect the press in any way.

Early in 1906 the Renfrow Briquette Machine Co. was incorporated under the laws of the State of Missouri with $1,000,000 capital; W. C. Renfrow, President; J. M. Smith, Secretary and Treasurer, and E. D. Mizner, Superintendent.

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This concern is a close corporation, and the company will not offer for sale any machines until after the Kansas City plant has proved successful. As far as can be learned, the construction of the Kansas City plant was brought about by the willingness of its President, Mr. J. H. Durkee, to accept a Renfrow machine without a guarantee, simply on the strength of the work done at the coal-testing plant. There are, of course, still some difficulties to be overcome, as is the case in the operation of any new plant, but in the main the mechanical operations of this plant are satisfactory, and the Renfrow Co. has been able to meet the required specifications. Financial difficulties have threatened the life of the plant under the present organization, but Governor Renfrow has affirmed that he will not allow this plant to fail for this reason. A contract has been signed to deliver one of the machines to a company at Detroit, but under the terms of the contract no date is fixed for the delivery of this machine, and no guarantee from the Renfrow Co. has been required. Governor Renfrow is also authority for the statement that the Detroit machine will not be delivered until after the Kansas City plant has been successfully operated and put on a commercial basis.

The Renfrow Briquette Machine Co. has no plant of its own, but has under consideration the establishment of a factory in St. Louis. All of the machines above mentioned were built in machine-shops under contract. The Kansas City machine was built by the Excelsior Tool & Machine Co., at East St. Louis, and all of the other machines by the Ramming Machine Co., of St. Louis.

North Dakota.

During 1905 ex-United States Senator W. D. Washburn, President of the Washburn Lignite Coal Co., erected a small plant at Minneapolis for experimental work in the briquetting of North Dakota lignite. The plant was too small to be operated successfully from a commercial standpoint. Several hundred tons of briquettes were made on this press without the use of a binder. The briquettes proved a satisfactory fuel for domestic purposes and for stationary boilers, but were not adapted to locomotive use, because the heavy exhaust-draft in the loco-

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motive has the effect of disintegrating the briquette before combustion and causes the throwing-off of large sparks. This small experimental plant represents the extent of Senator Washburn's efforts to briquette North Dakota lignites.

Mr. Robert L. Stewart, also of Minneapolis, who is interested in lignite-properties near Kenmare in Ward county, reports that he has been conducting a series of experiments with a view to the briquetting of this fuel, and as a result of his investigations the American Briquetting & Manufacturing Co. has been organized, which contemplates the construction, during the present year, of a briquetting-plant in North Dakota, convenient to the lignite-deposits, the plant to have a capacity of 1,000 tons of briquettes per day. Mr. Stewart says that the briquettes can be manufactured at a cost not to exceed $2 per ton f.o.b., this cost including the expense of mining the lignite and delivering it to the briquetting-plant.

Texas.

Three companies have been organized in Texas recently for the purpose of briquetting the lignites which occur in great abundance through the eastern part of that State. These are the International Compress Coal Co., of Houston; the American Lignite Briquette Co., of San Antonio; and the Eureka Briquette Co., of Rockdale. The plant of the last-mentioned company has been erected and is ready for operation at the time of the writing of this report, except for the fact that the drying-apparatus has been found too small, and the plant has been shut down pending the erection of a larger drier. The details of the plant have not been obtained.

The American Lignite Briquette Co., while incorporated at San Antonio, will locate its plant at Rockdale, and the plant will be operated in connection with the lignite-mines of J. J. Olsen & Son. The company has purchased a press made by the Klein Briquette Co., of St. Louis, and the plant will probably be in operation by the time this report is ready for distribution.

The International Compress Coal Co. has been negotiating for the construction of a plant, but no actual building had been begun at the time of writing this report. All of these plants expect to use asphaltic pitch made from heavy Texas oil.

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Florida.

In September, 1905, the Orlando Water & Light Co., of Orlando, completed the installation of a plant for the treatment and briquetting of peat, which occurs abundantly in the low-lying lands of Florida. The plant is located about 3 miles from Orlando, on the border of a peat-bog, from which its supply is drawn. As originally installed, this plant consisted of a macerating-machine or pug-mill, in which the fiber of the peat is entirely destroyed, and a brick-press. The briquettes, as they came from the press, were about the size of an ordinary building brick, and when dried in the sun shrunk to about one-fourth their former bulk and lost from 75 to 85 per cent. in weight. The briquetting-feature of the plant was abandoned in the summer of 1906, as it was found that this part of the work represented 75 per cent. of the total cost, and that a satisfactory fuel could be made without briquetting. The method of treatment at the present consists simply of "machining" the peat in the pug-mill and dumping it in masses of several hundred tons. As the peat dries it shrinks and cracks into large irregularly rectangular blocks, which are broken off from the heap and stored. When thoroughly dried these blocks make a good hard fuel, which, it is stated, may be used for both locomotive and stationary boilers, for household purposes, and for the manufacture of gas. Tests of the machined peat for producer-gas at the U. S. Geological Survey coal-testing plant gave excellent results.[18]

The machine used at the Orlando plant was built by the Moore & Wyman Elevator & Machine Works, South Boston, Mass., under patents issued to the late T. H. Leavitt, of Boston.

[18] Report on Peat, by M. R. Campbell, *U. S. Geological Survey, Mineral Resources of the United States*, 1905, pp. 1319 to 1322.

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Pure Coal as a Basis for the Comparison of Bituminous Coals.

BY W. F. WHEELER,* URBANA, ILL.

(Toronto Meeting, July, 1907.)

In the study of the coals of Illinois now being carried on by the State Geological Survey, an attempt is being made to determine the most satisfactory basis of comparison between different coals. The following discussion is based upon work done for the Survey in the laboratories of the State University and under the general direction of Prof. S. W. Parr.

"Pure coal" has been defined by Mr. A. Bement[1] as ash- and moisture-free coal, and the use of this "pure coal" as a basis for comparing different bituminous coals has lately been much discussed among engineers in the middle West. By some the B.t.u. of the pure coal is being used to check calorimetric results, and to calculate the calorific value of different coal-samples from the same seam, for which purposes it is necessary to assume—first, that the composition and calorific value of pure coal from different parts of the same coal-seam are uniform, and, second, that these values remain constant after the coal is mined, both of which assumptions there are reasons to believe are inconsistent with the facts. Certain variables are included in pure coal, as ordinarily defined, and disregard of them may lead to serious error. In any event, where delicate shades of difference are involved, no sure interpretation of results can be made, unless these variables be eliminated.

As a basis of comparison between coals, ash- and moisture-free coal is much to be preferred to "dry coal," and still more is it to be preferred to "moist coal," as ordinarily reported in analyses. There are, however, two constituents other than ash

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* Assistant Chemist, State Geological Survey of Illinois.

[1] *Journal of the American Chemical Society*, vol. xxviii., pp. 632 to 639 (1906); *State Geological Survey of Illinois, Bulletin* No. 3, pp. 23 to 25 (1906).

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and moisture which are similar in effect to them, and which constitute, therefore, variables. The first of these constituents, sulphur, is low in heat-value and variable in quantity, and figures for pure coal including it are misleading. The second constituent is the water of composition of the ash. It, too, is a variable for different ashes, and cannot be disregarded where close comparisons are to be made. The exclusion of the two larger variables, moisture and ash, does not justify calling ash- and moisture-free coal "pure coal," when it contains a widely varying amount of sulphur that is no more a part of it than the ash and moisture. Just because the mineral matter of the ash has sulphur combined with it, does not justify the inclusion of the sulphur as a part of the pure coal. An ideal pure coal should include the carbon, hydrogen, oxygen and nitrogen, and also the part of the sulphur that is not combined with the ash, that part which is in the coal in organic compounds. Practically, however, it is impossible to divide the sulphur between the ash and the pure coal, owing to difficulties arising in the determination of the organic and inorganic sulphur. When we know only the total quantity of sulphur in the coal, it is desirable to consider it all to be combined with iron as pyrite, and to discard it entirely, along with the ash and moisture. In thus considering the sulphur to be combined as pyrite, we must, in correcting for it, first correct the ash for the oxygen which is added to the iron in it to replace the sulphur. $2FeS_2 + 11 O = Fe_2O_3 + 4SO_2$. Three parts by weight of oxygen replaces eight parts of sulphur. This ratio gives us the easiest way of correcting the ash. Three-eighths of the sulphur subtracted from the ash will reduce the iron in it to the condition in which it was weighed in the coal. However, as we wish to discard the ash, moisture and sulphur, it is found more convenient to take the uncorrected ash plus the moisture and five-eighths of the sulphur from 100 per cent. and call the remainder sulphur-free pure coal. It will be easily seen that the method just given for calculating the percentage of pure coal in the moist coal will give the same result as is obtained by taking the corrected ash (the ash minus three-eighths of the sulphur), plus the total moisture and sulphur, from 100 per cent. The heat due to the sulphur in pyrite should not be credited to the pure coal, and is, therefore, to be deducted from it. According to

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calculation,[2] this amounts to 40.5 B.t.u. for each per cent. of sulphur. The organic sulphur in any coal-seam is probably a fairly uniform proportion of the pure coal, and any variation or error caused by the proposed consideration of it as pyrite will be small and constant in nature, and can, therefore, be neglected in so far as it affects the use of pure coal as a basis for comparing bituminous coals of ordinary sulphur-content. Variations of as much as 300 B.t.u. in the heating-value of the ash- and moisture-free coal can be traced directly to the variations in the sulphur-content.

Table I. shows, for six samples of Illinois and Indiana coal, calorific efficiency in B.t.u. of the "pure coal" corrected for sulphur, and also of the same coal as ordinarily considered (ash- and moisture-free). These were commercial samples, taken in the Chicago market by Mr. E. H. Taylor, of the Fuel Engineering Co. Part of each was floated in a zinc sulphate solution (sp. gr., 1.35), in order to remove a portion of the sulphur and ash, and thus get more nearly a pure coal to work with. The agreement between the two portions of the same sample (floated and original) is seen to be very much better where the sulphur is excluded. Ash- and sulphur-values are given in per cent. of dry coal.

The lack of agreement in some of the samples after correction for sulphur may be accounted for by the second variable (the water chemically combined with the ash). It will be noted that in only one out of the six pairs of samples is the B.t.u. of the pure coal higher for the sample with the high ash than for the one low in ash, and in that case the difference in ash is small. The suggested explanation is, that part, at least, of the ash is fire-clay, having chemically-combined water which is lost when the ash is ignited, but is not lost in drying. This variation is not due to the method of calculating the pure coal, but it is due to an analytical difficulty in determining the ash, and this difficulty is one that it is almost impossible to overcome. It affects the ultimate analysis, the water of the ash appearing there as oxygen and hydrogen; and in the proximate analysis it appears as volatile matter. For ordinary purposes it does not cause any discrepancies of importance. It is only so much more water in

[2] Report of the St. Louis Fuel-Testing Plant for 1904.

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TABLE I.—*Variation in Calorific Efficiency of Pure Coal with Differences in Ash.*

Sample No.	Kind of Coal.	Sulphur.		Ash, Uncorrected for Pyrite.		Corrected Pure Coal, Ash, Water and Pyrite Free.		Pure Coal, Ash and Water Free.	
		Original Coal.	Floated Coal.	Original Coal.	Floated Coal.	Original Coal.	Floated Coal.	Original Coal.	Floated Coal.
		Per Ct.	Per Ct.	Per Ct.	Per Ct.	B.t.u.	B.t.u.	B.t.u.	B.t.u.
1	Sangamon Co., Ill., Pawnee lump.....	5.99	...	11.66	...	14,319	...	13,987	...
2	Same as No. 1, floated	...	3.20	...	6.12	...	14,335	...	14,164
3	Sangamon Co., Ill., Latham screenings	4.25	...	18.21	...	14,285	...	14,031	...
4	Same as No. 3, floated	...	2.95	...	8.13	...	14,356	...	14,192
5	Williamson Co., Ill., Marion No. 5, washed nut.....	1.86	...	12.83	...	14,471	...	14,361	...
6	Same as No. 5, floated	...	1.39	...	4.01	...	14,585	...	14,509
7	La Salle Co., Ill., washed screenings	3.43	...	10.05	...	14,517	...	14,316	...
8	Same as No. 7, floated	...	2.33	...	3.94	...	14,619	...	14,487
9	Vigo Co., Ind., Miami nut.....	7.62	...	16.21	...	14,653	...	14,170	...
10	Same as No. 9, floated	...	3.08	...	4.27	...	14,653	...	14,478
11	Sullivan Co., Ind., Reliance lump.....	3.37	...	6.11	...	14,738	...	14,451	...
12	Same as No. 11, floated	...	1.29	...	2.53	...	14,709	...	14,624

the fuel, and it makes little difference for practical purposes whether that water was originally combined with the ash or with the coal. In extreme cases it might make a difference of as much as 2 or 3 per cent. in the ash-factor, but that would be only in the case of a low-grade coal very high in ash. In a coal containing 10 per cent. of fire-clay, having 10 per cent. of combined water, the difference in the ash, if determined, would be 1 per cent., and its effect, when calculated to the pure coal, would be to reduce the B.t.u. on that basis by about 1.1 per cent. Ten per cent. is not an unusual amount of combined water for a fire-clay. Different clays contain chemically-combined water varying from 5 to 12 per cent., and the average for Illinois fire-clays is somewhere near 8 per cent.

In an effort to determine the effect of variable ash on the B.t.u. of pure coal, eight samples from different mines produc-

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ing Illinois "No. 7" coal were each separated into two portions, one of which floated and the other sank in a zinc sulphate solution of a sp. gr. of 1.30. The great variation in ash thus secured in identically the same coal accentuated any variations that might in any way be due to the ash. In all of the eight samples, the B.t.u. of the coal which sank was found to be materially lower than that of the coal which floated. The close agreement between the difference in the B.t.u. of the sulphur-free pure coal and the difference in ash in the dry coal is shown in Table II.

TABLE II.—*Combined Water in the Ash, Calculated from the Difference Between the Calorific Efficiency of the Sulphur-Free Pure Coal in the Float- and Sink-Coal.*

Survey No. of Sample.	Comparison of Light and Heavy Portions of Same Coal.				Difference between Float- and Sink-Coal.		Difference in Weight of Ash in Raw Coal and in Cinder, due to Water of Composition, Expressed in Percentage of Raw Coal.	Difference in Weight of Ash Expressed in Percentage of Cinder.	Water of Composition of Ash Expressed in Percentage of Hydrated Ash.
	Float-Coal.		Sink-Coal.						
	B.t.u.	Ash.	B.t.u.	Ash.	B.t.u.	Ash.			
		Per Ct.		Per Ct.		Per Ct.	Per Cent.	Per Ct.	Per Ct.
104	14,462	...	14,227	...	235	...	.....	...	...
	...	4.64	...	18.00	...	13.36	1.3	9.7	8.8
105	14,510	...	14,271	...	239	...	.....	...	...
	...	3.83	...	14.43	...	10.60	1.4	13.2	11.7
106	14,414	...	14,096	...	318	...	.....	...	...
	...	4.22	...	22.17	...	17.95	1.7	9.5	8.7
107	14,404	...	14,343	...	61	...	.....	...	...
	...	2.56	...	9.52	...	6.96	0.4	5.8	5.5
108	14,587	...	14,374	...	213	...	.....	...	...
	...	4.08	...	17.75	...	13.67	1.2	8.8	8.1
109	14,657	...	14,434	...	223	...	.....	...	...
	...	4.34	...	18.28	...	13.94	1.2	8.6	7.9
112	14,641	...	14,558	...	83	...	.....	...	...
	...	3.40	...	10.89	...	7.49	0.5	6.7	6.3
113	14,718	...	14,602	...	116	...	.....	...	...
	...	3.23	...	13.41	...	10.18	0.7	6.9	6.5

On the assumption that the difference in calorific value was due to water combined with the ash and not to any variation in the sulphur-free pure coal in the two portions of the sample, the amount of water was calculated which would have to be com-

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bined with the ash to entirely account for the difference in the B.t.u. of the sulphur-free pure coal. The amount of water thus calculated and shown in Table II. agrees remarkably well with that ordinarily found in fire-clays and shales, when the possible variations due to experimental error are considered. In the mines from which the coals of Table II. were taken, the floor is a fire-clay and the roof is a shale very similar in appearance to the floor. There is always present, too, a "blue band" of shale from about 0.5 to 2 in. thick. On the supposition that the composition of the ash in the coal was similar to that of some of these materials, analyses were made of the material composing the floor and the roof, and also of the thin parting of shale in the mine supplying samples Nos. 108 and 109.

	Water of Composition. Per Cent.	Carbon. Per Cent.
Floor,	5.62	Not determined.
Roof,	4.20	Not determined.
Shale parting,	7.51	5.74
Calculated for ash in No. 108,	8.1	
Calculated for ash in No. 109,	7.9	

The water of composition was determined by igniting strongly on the blast lamp after drying for 1.5 hours at 105° C. to remove all moisture.

The agreement between the water of composition of the floor and "blue band" and that calculated for samples Nos. 108 and 109 is so close that it seems that we would be almost justified in using results obtained directly on such material as being representative of the ash in the coal. From the limited data at hand the indications are that the clay and shale under the coal and in partings distributed through it are very similar to the ash in the coal. The average water of composition calculated for the ash in samples Nos. 108 and 109 is 8 per cent., for the floor it is 5.6 per cent., and for the "blue band" 7.5 per cent. If there were a difference of 150 B.t.u. of the pure coal due to the 8 per cent. of water of composition, we would have reduced the difference to 45 B.t.u. by using 5.6 per cent. in correcting for it. If we had used 7.5 per cent. instead of 5.6 per cent. we would have reduced the discrepancy to 9.4 B.t.u. In none but extreme cases would such a discrepancy reach 150 B.t.u., and then it would be reduced to an amount that would

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not be very troublesome, by using the water of composition of the floor in correcting for it. It is doubtful if the water of composition of the ash, calculated from the analyses of "floated" and "sunk" samples, would be much, if any, more accurate than if it were obtained directly on the underlying material. The ash corrected by either of these methods would be represented in very nearly the condition in which it occurred in the coal.

In considering the possibility of using a constant B.t.u. for the sulphur-free pure coal from a given coal-seam for purposes of calculation and as a check on the calorimeter, we must take into account two other variables that have been overlooked heretofore, or have been thought to be of but minor importance. The first variable is the composition of the pure coal itself, which varies from top to bottom in the same bed, and is uniform for the whole bed only over limited areas. The second variable is due to the deterioration of the samples.

When the whole seam or corresponding sections of the seam are taken, the composition appears to be uniform for one mine, or for mines a mile or two apart, as in the case of samples Nos. 108 and 109, taken from the two mines of the Big Muddy Coal & Iron Co., east and west of Herrin respectively. The variation in composition of the different parts of the seam is so great as to make the coal indistinguishable from that of a number of other seams. The coal at the top of a seam is not the same as that at the bottom, and therefore samples from the same mine cannot be compared unless they represent the same portion of the coal-seam. The bottom coal in the Majestic mine, sample No. 106, has a calorific value of 14,338 B.t.u., while in the top coal, No. 109, it is 14,498, a difference of 160 B.t.u. for the two portions of the seam.

That there is an important variation in composition from point to point geographically is shown by the fact that eight samples of Illinois "No. 7" coal taken in Williamson and Franklin counties, Ill., show almost uniform decrease in the B.t.u. of the pure coal, going NW. from Marion. This difference (nearly 400 B.t.u.) must be considered.

Table III. shows the sulphur, ash and B.t.u. of the eight samples and Table IV. the falling off in their calorific value with increase in distance from Marion. Even though the de-

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crease shown is not very uniform, it is marked enough to afford positive proof that the composition of the pure coal varies gradually from place to place. We have no explanation to offer for this variation unless it is due to difference in the thickness and porosity of the material overlying the coal, or to the difference in the vegetation from which the coal was formed.

TABLE III.—*Analyses of Samples of Illinois "No. 7" Coal.*

S. G. S. No.	Lab. No.	Sulphur in Dry Coal.	Ash in Dry Coal.	Pure Coal Free from Ash, Moisture and Sulphur.
		Per Cent.	Per Cent.	B.t.u.
104	419	0.60	10.11	14,480
105	420	0.91	7.53	14,445
106	421	0.98	14.71	14,338
107	422	0.76	6.13	14,498
108	459	1.02	8.48	14,615
109	460	1.12	10.13	14,615
112	461	1.19	8.08	14,644
113	462	1.89	7.67	14,781

TABLE IV.—*Decrease in Calorific Efficiency of Sulphur-Free Pure Coal with Increase in Distance from Marion.*

S. G. S. No.	Locality.	Distance and Direction from Marion.	Total Decrease.	Decrease Per Mile.
			B.t.u.	B.t.u.
108	Herrin	5 Miles NW.	166	33
109	Herrin	7 Miles NW.	166	24
104 } 105 }	Ziegler	12 Miles NNW.	319	27
106 } 107 }	Duquoin	19 Miles NW.	395	21
112	Sesser	23 Miles N.	137	6

In fact, the variation in pure coal noted in analyses of "No. 7" coal is almost as great as is shown in the analyses of the

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different coal-seams of the State by the St. Louis fuel-testing plant of the United States Geological Survey.

Samples Nos. 104 and 105, taken from parts of the same mine a quarter of a mile apart, show practically the same B.t.u. for the sulphur-free pure coal. Samples Nos. 108 and 109, from mines two miles apart, show identically the same B.t.u., an exceptional agreement even for two analyses of the same sample. Unfortunately, no more analyses are at hand that have been made on fresh samples taken from the same coal-seam not more than two or three miles apart, except samples 106 and

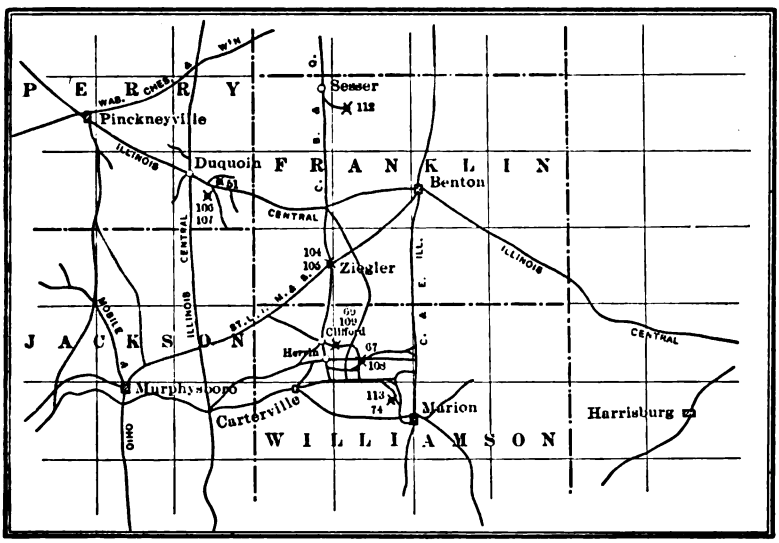

FIG. 1.—SKETCH-MAP SHOWING LOCATION OF COAL-MINES SAMPLED.

107, one of which represents the top part of the seam and the other the lower part, and they are therefore not comparable.

The map, Fig. 1, shows the location of the mines from which samples Nos. 51, 67, 74 and 106 to 113 were taken (all of Illinois "No. 7" coal). The location of the mines from which they were collected is marked on the map with an "X" and with the sample-number. All samples, except Nos. 106 and 107, represent complete sections of the working-face of the mine. No. 106 represents only the bottom coal, here 66 in. thick, while No. 107 represents the top coal, 30 in. thick, in an adjacent room. The complete list of samples discussed in this paper is given in Table V.

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TABLE V.—*Description of Coal Samples.*

Sample No.	Name of Mine.	Location.	Date.	Collector.
			1906	
8	Sangamon Mine.	Springfield, Ill.	May 1	F. F. Grout.
9	Sangamon Mine.	Springfield, Ill.	May 5	L. J. Rutledge and H. F. Bain.
51	Paradise Coal & Coke Co.	Duquoin, Ill.	May 21	F. F. Grout.
67	Chicago & Carterville Coal Co.	Herrin, Ill.	May 31	F. F. Grout.
69	B. Muddy Coal & Iron Co., No. 8.	3.5 miles N. of Clifford, Ill.	June 1	F. F. Grout.
74	Peabody Coal Co., No. 3.	3 miles NW. of Marion, Ill.	June 26	F. F. Grout.
76	Kelly Coal Co., No. 4.	Westville, Ill.	May 6	Tom Moses.
77	Kelly Coal Co., Himrod Mine.	Himrod, Ill.	May 17	Tom Moses.
			1907	
104	Ziegler Coal Co.	Ziegler, Ill.	April 16	H. F. Bain.
105	Ziegler Coal Co.	Ziegler, Ill.	April 16	H. F. Bain.
106	Majestic Mine.	3 miles south of Duquoin, Ill.	April 17	H. F. Bain.
107	Majestic Mine.	3 miles South of Duquoin, Ill.	April 17	H. F. Bain.
108	Big M. Coal & Iron Co., No. 7.	Herrin, Ill.	April 18	H. F. Bain.
109	Big M. Coal & Iron Co., No. 8.	Clifford, Ill.	April 18	H. F. Bain.
112	Keller Coal Co.	Sesser, Ill.	April 18	H. F. Bain.
113	Peabody, No. 3.	3 miles NW. of Marion, Ill.	April 18	H. F. Bain.
115	Sangamon Mine.	Springfield, Ill.	May 7	H. F. Bain.
117	Kelly Coal Co., No. 4.	Westville, Ill.	May 15	Bain and Moses.
118	Kelly Coal Co., Himrod M.	Himrod, Ill.	May 17	Bain and Moses.

In these studies a second important variable in the determination of pure coal is introduced by the deterioration of samples. The only analyses made use of as being representative of the coal as mined were those which were made within ten days of the time the samples were taken from the mine. That analyses of old samples do not represent the coal as mined, is indicated by the following circumstances:

It is known that combustible gas is liberated from moist laboratory-samples of coal when tightly sealed in glass jars and kept in a room of ordinary temperature, out of the direct sunlight. Evidence of this was afforded by a series of samples from all parts of Illinois studied in the course of our investigations. These were mine-samples, representing in each case the whole thickness of the coal-seam as mined, sampled down to about 700 g. at the mine and then placed in galvanized-iron cans having a screw top, and a piece of insulating-tape

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wound around the top to make it air-tight. When these samples were received at the laboratory they were immediately transferred to quart "Lightning" jars with the exception of the last 21, which were put into ordinary screw-top Mason jars. These samples stood for from six months to nearly a year, when they were opened to be air-dried and analyzed. A slight gas-pressure was noted in the first jar opened, and after that each one was tested with a lighted match; 21 out of 50 samples were in Mason jars and showed no pressure, and none of them contained gas enough to ignite. Of the 29 in "Lightning" jars, two had been opened previously and showed no gas. In one other, the gas was not inflammable, but was evidently carbon dioxide or nitrogen, as it put out the flame. The remaining 24 samples contained gas in varying amounts, each one igniting readily and burning with a blue flame from 0.5 to 6 in. in height. No analysis was made of the gas. The probable explanation of the fact that no gas was found in the Mason jars is that they were not gas-tight. A series of analyses of these old samples, compared with fresh ones from the same or adjacent mines, shows the extent of loss in the calorific value of the sulphur-free pure coal. Every case shows a considerable deterioration in the old samples.

The facts set forth in Table VI. show that analyses should always be made soon after taking the sample from the mine, and that even under the best of conditions (sealed in air-tight jars), coal of the type found in Illinois and Indiana will lose from 2 to 4 per cent. in heating-power in one year. How much it would lose if exposed to the weather and how rapidly such loss would take place is now being studied, but results are not yet available. Probably the loss would be greater for coal exposed to the air than for samples sealed in air-tight jars. That the deterioration was not due to the fine size of the sample is indicated by the fact that the loss was found to be practically the same for both portions of the same samples sealed, one ground to 100-mesh size, and the other to buckwheat size. The loss of volatile hydrocarbons will account for part, at least, of the loss in calorific value. Whether or not the gases given off are products resulting from the decomposition of the coal, or whether they were simply occluded gases, has so far not been determined.

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TABLE VI.—*Deterioration of Coal Samples.*

S. G. S. No.	Lab. No.	Pure Coal Free from Ash, Moisture and Sulphur.	Time Between Collection and Analysis.	Deterioration in Calorific Value.
		B.t.u.		Per Cent.
51	307	14,116	1 year.	1.9
106–7	421–2	14,386	10 days.	.....
67	323	14,321	1 year.	2.0
108	459	14,615	10 days.	.....
69	325	14,213	1 year.	2.7
109	460	14,615	10 days.	.....
74	330	14,335	1 year.	3.0
113	462	14,781	10 days.	.....
8	81	13,940	7 months.	4.3
9	82	14,100	7 months.	3.2
116	540	14,567	1 week.	.....
76	332	14,054	1 year.	2.8
117	557	14,450	5 days.	.....
77	333	14,087	1 year.	3.3
118	558	14,564	5 days.	.....

From the foregoing it seems clear that when delicate distinctions are to be made pure coal will furnish a better basis for comparison than any basis now in use; provided, of course, that correction be made for the sulphur and chemically-combined water in the ash. When these two factors are disregarded, the variations caused by them will be of sufficient magnitude to materially lessen the value that might otherwise attach to the pure-coal idea. It must not be forgotten, too, that any comparisons upon the pure-coal basis must take into account the fact that pure coal from one part of a seam is not the same as pure coal from another part of the same seam, there being a considerable variation, both vertically and horizontally. Probably the use for which pure coal is best adapted is for determining the extent of alteration in specific lots of coal in storage, but it is not without value for some of the other purposes mentioned. The calorific value of the sulphur-free pure coal may prove to be one of the most useful factors in classifying coals.

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The Production of Converter-Matte from Copper-Concentrates by Pot-Roasting and Smelting.

BY GEORGE A. PACKARD, BOSTON, MASS.

(Toronto Meeting, July, 1907.)

THE experiments here described were made under my supervision while temporarily acting as head of the Department of Metallurgy at the Missouri School of Mines, at Rolla. The work was done by Messrs. W. E. Brown, W. C. Richards, and F. L. L. Wilson, and the description of the results forms a portion of a thesis presented for the degree of Bachelor of Science. These results are submitted because I have seen no description of the application of pot-roasting to the treatment of a copper-concentrate.

This work followed that already described in the discussion of the Lime-Roasting of a Galena-Concentrate,[1] and was similarly occasioned by the lack of a furnace of the reverberatory type in which the copper-concentrates could be smelted. The material for the tests was a mixture of three lots available. The first consisted of 8-mm. jig-concentrates and the second of 0.5-mm. table-concentrates, from the San Juan district, Colo. Both contained pyrite, chalcopyrite, and galena. The third was a lot of copper-ore, found in the laboratory, which had been crushed through 8-mm. screen, and had at some previous time, before the laboratory roasting-furnace was torn down, received a partial roasting. The analyses of these ores are given in Table I.

In order to determine the applicability of pot-roasting as to each lot of ore, and the effect of varying proportions of lime, six preliminary tests were made in a "size N" Battersea crucible, 9 in. high by 6 in. in diameter at top, with a $\frac{3}{8}$-in. hole bored through the bottom. The crucible was set in the upturned elbow of a 2-in. pipe, the joint being luted with clay; and air was obtained from a receiver supplied by a compressor. The

[1] Discussion of paper of H. O. Hofman and others, this volume, p. 935.

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TABLE I.—*Analyses of Materials Treated.*

	Jig-Concentrates. Per Cent.	Table-Concentrates. Per Cent.	Partly Roasted Ore. Per Cent.
Fe, ...	31.97	36.60	27.20
Cu, ...	6.32	4.65	8.34
S, ...	38.85	44.10	6.48
Pb, ...	1.25	1.04	.....
CaO, ...	0.73	0.63	3.91
SiO_2, ...	17.42	10.32	12.48
Al_2O_3, ...	4.93	3.72	15.05
Au—oz. per ton, . . .	0.03	0.16	trace
Ag—oz. per ton, . . .	7.47	6.25	1.0

limestone, crushed through 8 mm., was mixed with the ore, and the mixture was well moistened before charging. No effort was made in these experiments to obtain a mixture corresponding to any definite slag-formula. The results obtained are given in Table II.

TABLE II.—*Crucible Experiments.*

Experiment No.	1.	2.	3.	4.	5.	6.
Weight, table-concentrates, . .	.....	1,000 g.	500 g.	300 g.	300 g.	300 g.
Weight, jig-concentrates, . .	1,000 g.	.....	500 g.	300 g.	300 g.	300 g.
Weight, roasted ore, . . .	.....	.....	.....	400 g.	400 g.	400 g.
Weight, limestone,	200 g.	200 g.	200 g.	100 g.	200 g.	300 g.
Total weight after roast, . .	838 g.	.....	898 g.	847 g.	968 g.	1,024 g.
Sulphur—						
Before, per cent.,	32.37	.....	34.56	24.98	22.89	21.14
After, per cent., .	5.16	.....	15.09	10.78	10.67	7.55
Eliminated, per cent., . .	88.89	.....	67.32	66.77	62.41	71.86
Blast, in inches of water, . . .	3 to 4	2 to 3	2 to 4	2 to 4	4	4
Time, in minutes,	69	90	40	35	34	34

Experiment No. 1, in which jig-concentrates only, mixed with 20 per cent. of limestone, were roasted, gave the best results, both in sulphur eliminated and in condition of product, which was quite solidly sintered, with but little loose, poorly roasted ore on top. The table-concentrates treated alone in No. 2 sintered very little; and it was evident that the method would not be applicable to such fine material unless it were mixed with coarser concentrates. Even then its presence in

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large proportions was detrimental, as shown by the high percentage of sulphur in the sintered product after roasting in experiment No. 3.

In Nos. 4, 5 and 6, a mixture of the ores in about the proportion available was used, and the lime was varied from 10 to 20 and 30 per cent. While No. 6, containing 30 per cent. of limestone, shows the smallest amount of sulphur after roasting, No. 4 seemed to be a little more completely sintered; and, since less lime was necessary, the proportions of No. 4 were used for roasting in the large pot.

This pot was of $\frac{1}{16}$-in. iron, 25 in. in diameter at top, 15 in. at bottom, and 23 in. high, with a circular $\frac{1}{8}$-in. sheet-iron grate, 17 in. in diameter, 5.5 in. above the bottom. This grate had $\frac{3}{8}$-in. holes at 1.25-in. centers. The pot was set on a brick foundation having an opening in the center 9.5 in. square, into which air was delivered by the pipe from the receiver, the pressure being indicated by a water-gauge and regulated by a valve in the pipe.

The remainder of the ore, consisting of 253.5 lb. of partly roasted ore, 174.5 lb. of table-concentrates, and 158.5 lb. of jig-concentrates, was then mixed together, and 63 lb. of limestone was added. The mass, thoroughly moistened, was charged into the pot on top of a small amount of glowing charcoal, which rested on a quantity of coarse limestone sufficient to prevent the fines from falling through the holes in the grate. It was treated in three charges: the unsintered fines from the first two, plus the fines formed in crushing these, being added to the third. The weights, air-pressure, etc., are shown in Table III.

TABLE III.—*Results of Pot-Roasting.*

	Weight of Mixture. Lb.	Time. Hr.	Pressure in Inches of Water.
Charge No. 1, ...	200	5.75	10 to 14
Charge No. 2, ...	280	7	8 to 14
Charge No. 3, ...	308	8.25	8 to 14
Total weight of ore and limestone, ...			649.5 lb.
Total weight of product, ...			534.5 lb.

This shows a total loss in weight of 115 lb., including sulphur, carbon dioxide, and fines blown out by the blast, the latter being very small in amount. The product was well sintered,

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practically constituting, in fact, a single lump, aside from about an inch of partly roasted material on top. The sintered portion was somewhat porous, though in places there were fused sulphides, resembling the heap-matte obtained in the heap-roasting of lump copper-ore when too strong a draft of air is admitted.

The entire product, including the fines from the last charge, was sampled down and analyzed, showing 16.17 per cent. of SiO_2, and 30.30 of Fe, 7.59 of Cu, 13.16 of Al_2O_3, 9.04 of CaO and 9.14 of S. This shows the elimination of 69 per cent. of the sulphur present. The gold, silver and lead were not determined.

This product, mixed with 13 per cent. of its weight of sandstone (nearly pure silica), and a small amount of old slag, and smelted in a small blast-furnace having a diameter of 26 in. at the tuyeres, yielded a matte carrying 32.11 per cent. of Cu, and 19.24 oz. of Ag and 0.12 oz. Au per ton.

This matte is low in copper for immediate converting, but the concentrates were probably somewhat lower in copper than would ordinarily be the case. The grade of the matte would probably also be increased in ordinary practice by the effect of the greater height of the furnace and the greater volume and pressure of blast, under which conditions more sulphur would be burned off. In our smelting of this small amount of ore the top of the ore-column was never more than 3 ft. above the tuyeres, and the blast-pressure never exceeded 5 in. of water. It also seems probable that the amount of sulphur in the sintered product might be decreased by the regulation of the blast so as to prevent the formation of the heap-matte in the pot. Another factor which here increased the sulphur going to the blast-furnace, and which would be eliminated in practice, was the inclusion of the partly roasted fines from the last pot-roast in the blast-furnace charge. These would ordinarily go to the next pot-charge. It seems possible to reduce the amount of these by careful attention, and by pressing them down with a heavy rabble when the charge is becoming red on top.

The preliminary experiments indicate another important factor in the amount of sulphur eliminated—namely, the coarseness of the material. Thus, the 0.5-mm. concentrates, treated alone, sintered but little; and the students did not consider the prod.

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uct worth working with. When mixed with an equal quantity of 8-mm. concentrates, the sulphur elimination was 67 per cent.; while the coarse concentrates, treated alone, gave up 88 per cent. of their sulphur, and sintered best of all.

The product obtained is, after crushing, in excellent condition for treatment in the blast-furnace; and, while it would doubtless not be economical to bring the sulphur down to a point representing a high-grade copper-matte, the production of a satisfactory matte for converting appears to be practicable. Where fuel is expensive, this method would apparently be cheaper than roasting and smelting the concentrates in a reverberatory, prior to converting.

The term "pot-roasting" has been preferred to "lime-roasting," because it has been shown with matte-roasting, in the West, that the presence of lime is not, though a certain amount of silica apparently is, necessary to the formation of a sintered product. (I question if the same may not be true in the roasting of galena.)

In the case of these concentrates, the operation seemed to present conditions analogous to those existing at the top of a furnace in which pyritic smelting is done. At first, dense yellow sulphur-fumes were evolved; and the sublimed sulphur collected on the cold objects in the vicinity. After a short time, the white sulphur dioxide fumes became more prominent, and continued until the end of the roast.

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The Effect of High Litharge in the Crucible-Assay for Silver.

BY RICHARD W. LODGE, BOSTON, MASS.

(Toronto Meeting, July, 1907.)

In the crucible-method of assaying ores for silver a certain amount of litharge is essential to supply sufficient lead to collect the precious metals. The object of this paper is to point out that the use of a large excess of litharge in the assay of some ores will give results for silver that are uneven as well as low. So far as I know, however, an excess of litharge does not affect the results obtained in the crucible-assay of ores for gold.

The main reasons for using, in the crucible-assay, much more litharge than is required to give the necessary lead button are: 1, its action as a flux; 2, its action as a desulphurizer; and 3, its action as an oxidizer, especially on metals like copper and nickel, whereby they are forced into the slag as oxides and thus prevented from passing into the lead button.

Hence, by the use of much litharge in the crucible-assay more ore can often be taken than in the scorification-process, and a lead button obtained which can possibly be cupelled at once or after a single scorification. The method is specially advantageous with an ore carrying much copper or similar impurity and poor in silver, when the assayer does not wish to resort to a wet-analysis for the determination of the silver.

It is well known that some ores give better results when assayed by the scorification-method, while others give better results by the crucible-method.

For several years I have noticed that, when much litharge was used with certain sulphide and arsenical ores, the results were considerably lower than when the scorification-method was used.

This fact was more forcibly brought out in connection with certain work carried on this year by Messrs. H. A. Frame and

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F. C. Jaccard, students at the Massachusetts Institute of Technology, to ascertain the best method of assaying the rich arsenical nickel- and cobalt-ores from Ontario, Canada.

The following are some results obtained: Ore No. 2,687–2 had a reducing-power of 4.2 and contained Ni, 12.92; Co, 10.92; and As, 46 per cent. The minerals noticed in the ore were niccolite, smaltite, erythrite, cobaltite, and arsenopyrite.

Charge for the Crucible.					Results.			
Ore.	Sodium Bicarbonate.	Borax-glass.	Litharge.	Argols. (R. P. = 11)	Lead Button.	Silver Found.		Silver in Slag.
	g.	g.	g.	g.	g.	g.	Oz. per Ton.	
1/10 A. T.	10	5	30	1½	23	0.11615	2323	Not assayed.
1/10 A. T.	10	10.	35	1½	23	0.11648	2329.6	Not assayed.
1/10 A. T.	10	5	80	1½	28	0.11272	2254.4	52.4 oz.

Ore No. 2,687–6 consisted chiefly of smaltite, erythrite, niccolite, and arsenopyrite; R. P. = 4.06; Ni, 3.94; Co, 11.25; and As, 59.7 per cent.

Ore.	Sodium Bicarbonate.	Borax-glass.	Litharge.	Argols.	Lead Button.	Silver.
	g.	g.	g.	g.	g.	Oz. per Ton.
1/10 A. T.	10	10	35	1.5	22	258.2
1/10 A. T.	10	10	80	1.5	21	241.2

The same ore assayed after amalgamation:

1/10 A. T.	30	10	30	1.5	23	229.8
1/10 A. T.	10	10	35	1.5	23	230.8
1/10 A. T.	10	10	80	1.5	23	220.2

Subsequent to the above tests I made further investigations on the ores tested as well as on other ores and obtained the following results: Ore No. 2,687–2; R. P. = 4.2; Ni, 12.92; Co, 10.92; and As, 46 per cent.

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Scorification-Method.—Charge.				Results.	
Ore.	Borax-glass.	Lead.	SiO_2.	Lead Button.	Silver.
	g.	g.	g.	g.	Oz. per Ton.
1/20 A. T.	3	65	1.5	11	2330.4
1/20 A. T.	5	65	1.5	10	2338

Crucible-Method.						
Ore.	Sodium Bicarbonate.	Borax-glass.	Litharge.	Argols. (R. P. = 11)	Lead Button.	Silver.
	g.	g.	g.	g.	g.	Oz. per Ton.
1/20 A. T.	10	6	35	1.5	22	2333.2
1/20 A. T.	10	6	35	2	30	2330.8
1/20 A. T.	10	6	80	1.5	23	2229.2
1/20 A. T.	10	6	100	1.5	25	2244.2

Ore A. Chiefly smaltite and niccolite with free silver, containing Ni, 5.06, and Co, 9.12 per cent.

Ore.	Sodium Bicarbonate.	Borax-glass.	Litharge.	Argols.	Lead.	Silver in Lead.	Silver in Slag.	Silver in Cupel.
	g.	g.	g.	g.	g.	Oz. per Ton.	Oz. per Ton.	Oz. per Ton.
1/20 A. T.	10	10	30	1.5	19	2051.4	9.6	34.0
1/20 A. T.	10	10	40	1.5	21	2056	.....	.....
1/20 A. T.	10	10	40	1.5	21	2050	.....	.....
1/20 A. T.	10	10	80	1.5	30	1968.6	.....	.....
1/20 A. T.	10	10	80	1.5	22	1944.6	135.2	35.0
1/20 A. T.	10	10	80	1.5	21	1984.8	70.2	34.6
1/20 A. T.	10	10	80	1.5	21	1914.8	.....	.....

Ore No. 2,703–1 consisted of smaltite, native bismuth and native silver in calcite; R. P. = 5; Ni, 0.3; Co, 8; and As, 55 per cent. The scorification-assay gave 404.8 oz. and the combination wet-and-dry analysis gave 403.7 oz. of silver per ton.

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Crucible-Method.

Ore.	Sodium Bicarbonate.	Borax-glass.	Litharge.	Argols.	Silver in Lead.	Silver in Slag.
	g.	g.	g.	g.	Oz. per Ton.	Oz. per Ton.
$\frac{1}{20}$ A. T.	10	6	35	1.5	402.6	Not assayed.
$\frac{1}{20}$ A. T.	10	6	80	1.5	388	Not assayed.
$\frac{1}{20}$ A. T.	10	6	100	1.5	392	11.8

The same ore after amalgamation gave, by the scorification-assay, 292 and 291.2 oz., while by the combination wet-and-dry analysis the result was 292.2 oz. of silver per ton.

Crucible-assays.

Ore.	Sodium Bicarbonate.	Borax-glass.	Litharge.	Argols.	Lead Button.	Silver.
	g.	g.	g.	g.	g.	Oz. per Ton.
$\frac{1}{20}$ A. T.	10	6	35	1.5	24	293
$\frac{1}{20}$ A. T.	20	6	35	1.5	23	292.8
$\frac{1}{20}$ A. T.	20	6	35	1.5	25	291.8
$\frac{1}{20}$ A. T.	10	6	60	1.5	24	288
$\frac{1}{20}$ A. T.	10	6	80	1.5	26	283.6
$\frac{1}{20}$ A. T.	10	6	100	1.5	26	279.8

I have a number of other examples, but these should be sufficient to illustrate the fact to which I wish to call attention. The ores used carried practically no gold.

The same lot of litharge was used in all the fusions, and the conditions under which the fusions were conducted were as nearly identical as possible. Heavy crystals of litharge were found on all the cupels.

Only $\frac{1}{20}$ A. T. of ore was used in the assays, because Messrs. Frame and Jaccard found that in case of ore No. 2,687–2, carrying 12.92 per cent. of nickel, if 5 g. of ore were taken and 35 g. of litharge were used, the resulting lead button would not cupel. If 5 g. of ore were taken and the litharge increased to 80 g., in order to slag the nickel, the button would cupel but the silver-results were low.

Some ores from the Cobalt district carry so large a percent-

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age of nickel that $\frac{1}{20}$ A. T. of ore is the limit that can be used in either scorification- or crucible-work, and low litharge is inapplicable in the latter method. The following ore serves as an illustration: Ore No. 2,703–2, niccolite (NiAs); R. P. = 5.3; Ni, 38.01; Co, 1.19; and As, 53.31 per cent.

Ore.	Sodium Bicarbonate.	Borax-glass.	Litharge.	SiO_2.	Argols.	Ratio of Litharge to Ni in Ore.	Lead.
	g.	g.	g.	g.	g.		
$\frac{1}{20}$ A. T.	15	10	120	5	1	214 to 1	24
$\frac{1}{10}$ A. T.	15	10	120	5	None.	107 to 1	25
$\frac{2}{10}$ A. T.	15	10	120	5	Nitre–1	54 to 1	25

The lead buttons from $\frac{1}{10}$ and $\frac{2}{10}$ A. T. of ore would not cupel, both cupels being covered with a thick film of green nickel oxide (NiO).

The lead button from $\frac{1}{20}$ A. T. of ore would just cupel, leaving the cupel stained green.

One of the advantages of the crucible-method is that it enables the assayer to use a larger amount of ore than in the scorification-method, but from the foregoing data there seems to be no advantage in this method over scorification on ores carrying much more than 10 per cent. of nickel. When, however, the ores, either siliceous or calcareous, are poor in silver and carry only a small percentage of nickel or similar impurity, the crucible-method, with low litharge, can be used to advantage.

Cobalt is much more readily slagged than nickel, especially in the presence of alkali and silica, and does not so readily pass into the lead button, therefore ores quite rich in cobalt can be assayed by the crucible-method, a considerable amount of ore be used and the litharge kept low.

From 80 to 90 per cent. of the cobalt, if a reasonable amount of ore is used, will pass into the slag in either scorification- or crucible-work.

It is a question with me as to the cause, in certain cases, of the uneven and low results in silver when high litharge is used in the crucible-assay, but apparently the silver passes in some way into the slag. At first I thought that, in the examples given, either the arsenic, the nickel or the combination of both was the cause, but this was not true of all ores. Some ores

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carrying considerable nickel, cobalt and arsenic gave good results as well as an ore carrying Co, 0.67; Ni, 0.44; and As, 0.6 per cent. in a gangue consisting of silica and calcite.

The question of high or low temperature does not seem to influence the slagging of the silver unless this takes place at some particular period of the fusion. Owing to this uncertainty as to the action of the litharge, I prefer the scorification-method for ores from the Cobalt district which carry much nickel, using $\frac{1}{20}$ or $\frac{1}{10}$ A. T. of ore, from 3 to 8 g. of borax-glass, high lead (65 g. or more), with some silica, and a medium high temperature. If obliged to use the crucible-method, I keep the litharge low and take such an amount of ore that both high litharge and nitre are avoided.

If all ores from this district are ground very fine, thus removing the greater part of the silver pellets, it is surprising how uniform are the results obtained by the scorification-method—something that I have not found by the crucible-method nor in some ores when using 0.5 A. T. for the combination wet-and-dry analysis.

It is hoped that this paper will bring out the experiences of some other assayers.

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Chronology of Lead-Mining in the United States.*

BY W. R. INGALLS, NEW YORK, N. Y.

(Toronto Meeting, July, 1907.)

THE following chronology presents the history of lead-mining in the United States in a brief form and is a useful reference in connection with the statistics of production:

1621.

Lead was mined and smelted near Falling Creek, Va., to supply the local demand for bullets and shot. This was the first mining and smelting of lead in what is now the United States.

1632.

In a report made on the minerals of New England, lead-ore is mentioned. (Bishop, I., 470.)

1650.

Supposed beginning of mining by Jesuits in Pima county, Ariz.

1651.

Grant of lead-mine at Middletown, Conn., to Gov. John Winthrop.

1682.

Lead-ore supposed to have been discovered in Wisconsin by Nicholas Perrot. (R. D. Irving, Mineral Resources of Wisconsin, *Trans.*, viii., 498.)

1700.

Discovery of lead in Missouri by Penicaut, one of Le Sueur's party. The same expedition discovered lead near the southern boundary of Wisconsin in August of the same year.

1712.

Grant by Louis XIV. of the Crozat patents, with special privileges respecting the discovery and operation of mines in the then territory of Louisiana. Little or no mining was done under this patent until about 1720.

1717.

Transfer of Crozat patents to the Mississippi Co., promoted by John Law, which prepared for active mining.

1719.

First attempt to mine and smelt lead in Missouri, made by Sieur de Lochon, in behalf of the Mississippi Co., near Meramec river. Results unsuccessful.

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* This paper was prepared for the Carnegie Institution, Washington, D. C., as a part of the industrial history of the United States, and is published here with the permission of Hon. Carroll D. Wright, director of that work.

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1720.

Philip Francis Renault, appointed director-general of the mines of the Mississippi Co. in 1719, arrived at Kaskaskia with 200 artisans and miners and 500 slaves, and sent out exploring parties from there, one of which discovered the deposits of Mine La Motte, in Madison county, the mine taking its name from M. La Motte, a mineralogist accompanying Renault, under whom it was operated.

1723.

Grant of Mine La Motte to Renault.

1724–6.

Discovery of lead at Old Mine and Mine Renault, north of Potosi, Washington county, Mo.

1730.

A company of German miners was sent out to the colony of New Netherlands by Baron Horsenclaver. These miners explored the Highlands and made many ventures in mining and smelting.

1731.

Failure of the Mississippi Co. and reversion of its charter to the Crown.

1738–1740.

About this time the Mine La Motte was considered public property, and the people in general were allowed to work at it. It furnished almost all the lead then exported from the Illinois (Moses Austin).

1740.

Operation of lead-mine near Northeast, Dutchess county, N. Y. The mine produced a small quantity of ore, but not profitably. The ore was sent to Bristol, England, and to Amsterdam.

1742.

Return of Renault to France, bringing to a close the first period of mining in Missouri.

1750.

Discovery of the Wythe mines, Virginia, by Col. Chiswell. Worked during the Revolutionary War, and afterwards intermittently up to 1838, and since then rather continuously.

1754.

Lead-ore was known to exist at Southampton, Mass., as early as this year, and lead was mined at Worcester, Mass. (Bishop, I., 493.)

1762.

Cession of Louisiana to Spain.

1763.

Discovery of Mine à Burton, at Potosi, Mo., and immediate inauguration of exploitation. This, together with Old Mine and Mine Renault, both near Potosi, and the Mine La Motte, were the principal mines worked during the next 30 years.

1765.

Development of lead-mine at Southampton, Mass. Work suspended by Revolutionary War and not resumed until 1809. Never became important.

1766.

Capt. John Carver found lead-ore in abundance at Blue Mound, Wis. The Indians knew of it, but did not know how to obtain the metal.

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1767.

Governor Clinton, of New York, directed attention to the existence of valuable veins of lead-ore in that colony, and stated that the British Government had leased a mine of argentiferous galena to Frederick Philipse. A large refinery of lead or of iron existed at Sing Sing prior to, or at the beginning of, the Revolution. (Bishop, I., 527, 533.)

1769.

Destruction of settlement at Mine La Motte by Chickasaw Indians and abandonment of the mine, which was not reopened until 1780 or 1782.

1778.

Operation of lead-mine near Birmingham, Blair county, Pa.; resumed in 1795, and again in 1864. Never important.

1788.

The first mining in the Wisconsin-Iowa region was done at Dubuque, Iowa, by Julien Dubuque, who received grant of a lead-mine from the Fox tribe of Indians. Dubuque worked this mine until his death (in 1809).

1789.

Tariff on lead fixed at 1c. per lb.

1795.

Discovery of the Mine à Lanye, about 16 miles SE. of Potosi, Mo.

1799.

Discovery of the Mine à Maneto, on Big river, St. François county, Mo.; also Mine La Platte, about two miles from Big river, near the SE. corner of Washington county.

Arrival in Missouri of Moses Austin from Wytheville, Va.; improvement of smelting methods, erection of shot-tower, and works for manufacture of sheet-lead.

1801.

Discovery of Mine à Joe, later called the Bogy Mine, on Big river, St. François county, Mo.

1803.

Discovery of Mine à Martin, near Potosi, and several other deposits in Washington county, Mo.

Louisiana purchased by the United States. Most of the French and Spanish concessions, when they had been continuously occupied, were confirmed by a commission.

1806.

Discovery of New Diggings, near Potosi, Mo., which for a few years produced about 1,000 tons of galena per annum.

Discovery of mines at Hazel Run, about five miles NE. of Bonne Terre, St. François county, Mo., which are said to have yielded 500 tons of lead in the first year.

1807.

Act of Congress reserving all lead-lands in territory of Louisiana and authorizing the Governor to grant three-year leases to discoverers at royalty of 10 per cent. of the product. (*American State Papers*, 2d ed., IV., pp. 526, 555.)

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1811.

Discovery of Shibboleth mines, near Cadet, Washington county, Mo., which in the first year are said to have yielded 2,500 tons of ore, equivalent to 1,563 tons of lead.

1812.

Tariff on lead raised from 1c. to 2c. per lb.

1814.

The Fourche à Courtois mines, at Palmer, Washington county, Mo., were discovered.

1816.

Tariff on lead fixed at 1c. per lb.

1820.

Lead-ore worked at Ellenville, N. Y., but with little success. Several veins worked there about 1854 and two Scotch hearths erected.

1821.

Attention attracted by explorers to the Wisconsin lead-region.

1824.

Import duty on lead raised from 1c. to 2c. per lb.

Discovery of Sandy mines, near Hillborough, Jefferson county, Mo., which soon became large producers.

Joseph Schutz discovered the Valle mines, seven miles north of Bonne Terre, St. François county, Mo.

1825.

Bisch's mine, near the Valle mines, was discovered.

1826.

Development of Eaton mine, near Madison, Carroll county, N. H. Vein leanly mineralized with blende and argentiferous galena.

1828.

Tariff on lead raised from 2c. to 3c. per lb.

1830.

Golconda mine in Franklin county, Mo., discovered.

1832.

Discovery of small veins of lead-ore near Lubec, Me., and beginning of development; results never successful.

Final withdrawal of the Indians from Wisconsin.

1834.

Discovery of Virginia mine, near St. Clair, Franklin county, Mo., extensive developments being immediately undertaken. Smelting begun in 1835.

In consequence of the large number of illegal entries, the miners and smelters of Missouri refused to pay royalties and the Government was unable to collect. (J. D. Whitney, *Metallic Wealth of the United States*, p. 405.)

1835.

Operation of lead-mines at Rossie, N. Y.; continued until 1840.

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1836.

Discovery of Washington, known later as Silver Hill, mine in Davidson county, N. C. Worked almost uninterruptedly until 1852. Reopened in 1855.

Erection at Webster, Washington county, Mo., of first Scotch hearth furnace in Missouri. (G. C. Swallow, *Report of Geological Survey of Missouri*, II., 59.)

1837.

Reed and Hoffman erected works for manufacture of white lead at St. Louis. Other works were erected soon afterward at the same place.

Operation of vein of lead-ore near Redbridge, N. Y.

1838.

Value of cerussite ore, "dry bone," first recognized in Missouri, leading to the erection of new furnaces and an increase of product. (James E. Mills, *Geological Report on the Mine La Motte Estate*, p. 47.)

1839.

Discovery of lead-ore at Rosiclare, Hardin county, Ill.

1846.

Operation of lead-mine near Shelburne, Coos county, N. H.

Tariff on lead reduced to 20 per cent. ad valorem.

1847.

Congress decided to sell the National lead-lands in the Mississippi valley.

1848.

The Avon mines, Ste. Genevieve county, Mo., produced eight tons of lead.

Mining of lead-ore begun two miles east of Joplin, Mo., by William Tingle.

1850.

Mining done on small scale near Phœnixville, Chester county, Pa.

Mining begun near Granby, Newton county, Mo., in which vicinity operations were well under way by 1857 and a large output was being made.

The air, or Drummond, furnace was first tried for lead-smelting in Newton county, Mo.

1851.

Mining begun on Center creek, near what was later called Minersville, now Oronogo, near Joplin, Mo.

Erection of first Scotch hearth furnace in southwestern Missouri, located near mouth of Cedar creek, Newton county.

Lead-smelting furnace erected on West Sugar Loaf creek, Ark., this being the first in that State.

1852.

Resumption of mining at Rossie, N. Y.

1853.

Resumption of mining at Ancram, Columbia county, N. Y.

1855.

The Mowry mine, south of Tucson, Ariz., purchased by Major Ewell and others.

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1857.

Tariff on lead reduced to 15 per cent. ad valorem.

Establishment of town of Granby, Mo., and erection of furnaces by Peter E. Blow and Ferdinand Kennett.

1858.

Discovery of rich vein of lead-ore at Guymard, N. Y.; operated until 1868 and later.

The Mowry mine passed into the hands of Lieut. Sylvester Mowry, who from this date until 1862 operated it on a considerable scale. This appears to be the first silver-lead mine west of the Rocky mountains to have been operated in an extensive way. The Confederate army is reported to have been supplied with some lead from this source.

1859.

Discovery of mines at Georgetown, Colo.

1861.

Tariff on lead raised to 1c. per lb., and later in the year to 1.5c. per lb.

Mine La Motte furnaces destroyed by United States Government, but soon rebuilt.

1862.

Plant of the Mowry mine, Arizona, destroyed by Federal troops.

1863.

First discoveries of argentiferous lead-ore in Little Cottonwood cañon, Utah.

Discovery of the Jordan mine, Bingham cañon, Utah.

Discovery of silver-lead mines at Castle Dome, Ariz., which, on account of Indian hostilities, were not actively worked until 1869.

1864.

First locations at Eureka, Nev., but no important developments were made until 1869, in which year the great silver-lead deposits were opened.

Organization of St. Joseph Lead Co., which purchased La Grave mines at Bonne Terre, Mo. Active operations begun in 1865.

Tariff on lead raised to 2c. per lb.

1865.

Organization of Granby Mining & Smelting Co. to work the mines at Granby, Mo.

Erection and operation of smelting-works at Argenta, Mont.; commonly credited as the beginning of silver-lead smelting in the United States.

1866.

Establishment of the Selby smelting- and refining-works at San Francisco, Cal.

1867.

Discovery of silver-lead ore in the Magdalena mountains, N. M.

Discovery of rich deposits of silver-ore at White Pine, Nev.; these were the first large bodies of silver-ore found in a limestone formation in the United States, and the information gained from them led directly to the discovery of the silver-lead deposits of Eureka soon afterward.

Smelting begun at Oreana, Nev.

1868.

The Emma mine, Little Cottonwood, Utah, was located in August of this year, but no large shipments were made until July, 1870.

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1869.

The junction of the Union Pacific and Central Pacific tracks was made at Promontory, Utah, May 10, 1869. The Utah Central railway was completed to Salt Lake City in December, 1869. The completion of the Pacific railways greatly stimulated prospecting along their lines, making available to market the lead in ores previously discovered in Utah and Nevada.

Development of silver-lead mines at Cerro Gordo, Cal.

Inauguration of diamond-drill prospecting by St. Joseph Lead Co. at Bonne Terre, Mo., and discovery of disseminated ore at depth of 120 ft.

The important silver-lead deposits of Eureka, Nev., began to be productive. The American practice of silver-lead smelting has been developed chiefly from the methods introduced in this district.

1870.

First important developments in the districts of Big and Little Cottonwoods, Bingham cañon, Stockton and Tintic, Utah.

The Miller mine in American Fork cañon was discovered, but was not worked extensively until 1871.

Construction of narrow-gauge railway-system in Colorado begun by Denver & Rio Grande Railway Co.

Discovery of lead-mines at Rosita, Colo.

The mines of Eureka, Nev., become large producers.

Beginning of shipments from the great Emma deposit, Little Cottonwood cañon, Utah. This was the principal source of lead in Utah up to 1875, when the deposit was exhausted.

Invention of the siphon tap for lead blast-furnaces by Albert Arents, and its application at Eureka, Nev.

Erection of lead-smelting works at Omaha, Neb.

Discovery of lead-ore in large quantity at Joplin, Mo., followed by the rapid development of that district.

1871.

The mines of Big and Little Cottonwood, Utah, made large shipments.

Discovery of silver-lead ore in Parley's Park district, now Park City, Utah.

Chicago an important smelting and refining center.

1872.

The Ontario vein, Park City, Utah, was located June 19.

Discovery of silver-ore at Georgetown, N. M.

Refining of lead begun at the Germania works, Salt Lake City, Utah.

E. Daggett installed cast-iron water-jackets at the Winnamuck smelting-works, Utah, these being the first water-jackets in Nevada-Utah smelting practice and the first cast-iron jackets employed anywhere.

Discovery of lead-ore in Cherokee county, Kan.

Tariff on lead reduced to 1.8c. per lb.

1873.

Discovery of silver-lead mines in Wood river district of Idaho.

The United States, by Act of Congress, Feb. 12, discontinued the coinage of silver dollars. This Act did not demonetize silver in words, although it did so in effect. The silver dollar was not named in it. Precisely what the Act did was to authorize the coinage of silver half-dollars, quarter-dollars and dimes below standard weight, and of a new silver coin for Asiatic commerce, of standard

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weight, to be called the "trade dollar," and to prohibit these coins from being legal tender for more than five dollars in any one payment. The German Government, by Act of July 9, provided for the retirement of its silver coins and the sale of the bullion. By a Treasury order, Sept. 6, France limited the amount of silver to be accepted by its mint. These actions, which were soon afterward followed by similar ones in other countries, were closely involved with the silver question, and the decline in the value of silver, which began at this time, culminating in the crisis following the closing of the Indian mints to the private coinage of rupees, June 26, 1893, had a powerful effect on the silver-lead industry.

1874.

Early in this year, argentiferous lead-carbonate ore was found on Iron Hill, Leadville, Colo., and the Lime and Rock claims were located.

Discovery of silver-lead ore at Darwin, Inyo county, Cal.

Installation of dust-chambers at several Western lead-smelting works and adoption of methods for further treatment of matte.

1875.

Discovery of Horn Silver mine, Frisco, Utah.

Mining was begun at Webb City, Mo.

Tariff on lead raised to 2c. per lb.

1876.

First shipments from Leadville, Colo.

Mining was begun at Carterville, Mo.

Investigations by Anton Eilers and others determined the correct principles in preparing charges of ore for smelting, a development of great economic importance.

1877.

The Bassick mine, near Silver Cliff, Colo., began to show evidence of value.

First smelting-works erected at Leadville, Colo.

1878.

Discovery of the silver-lead deposits of Sierra Mojada, Coahuila, Mexico.

Great excitement at Leadville, Colo., where many new discoveries were made. The output of ore began to be large.

Mines of Hecla Consolidated Mining Co., at Glendale, Mont., became productive.

Discovery of lead-carbonate ore in the eastern part of Gunnison county, Colo.

Blast-furnaces substituted for reverberatory at works of St. Joseph Lead Co.

The American Pig Lead Association, an alliance of the principal lead-miners and lead-smelters of the United States, was formed to maintain the price of lead at minimum of 4c. per lb. The attempt failed.

Introduction of Lewis & Bartlett process at Lone Elm smeltery, Joplin, Mo.; this was the first application of cloth-filtration of fume in the metallurgy of lead.

Desloge mill and furnace, adjacent to works of St. Joseph Lead Co., Bonne Terre, Mo., put in operation.

First location made at Tombstone, Ariz.

1879.

First important discoveries in the Wood river district of Idaho. Ore had been known to exist in this district since 1873, but developments were checked by Indian troubles and not actually begun until 1880. The district became productive in 1881, making the first important output of lead in Idaho.

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Discovery of lead-carbonate ore at Rico, Colo.; also at Red Cliff, Colo., and at Kokomo, Colo. Considerable excitement in the Gunnison country, Colo.

Discovery of promising deposits of silver-ore at Aspen, Colo., and in the San Juan region in the southwestern part of the same State.

Lead-mines discovered at Barker, Meagher county, Mont.

1880.

Completion of the Southern Pacific railway through Arizona.

The Denver & Rio Grande railway reached Leadville, Colo.

Discovery of Silver Valley mine, Davidson county, N. C.

Strike of miners at Leadville, Colo.

Discovery of lead-ore at Robinson and Kokomo, Colo.

Excitement in the Gunnison district of Colorado, which did not, however, materialize into developments of great importance.

St. Louis & San Francisco, Missouri Pacific, and Kansas City, Fort Scott & Memphis railways extended into Joplin district, Mo.

1881.

Establishment of smeltery at Socorro, N. M., for treatment of ores of Socorro and Magdalena.

1882.

Discovery of the Viola mine at Nicholia, Idaho.

Red Cliff, Colo., began to make a considerable output.

1883.

Maximum output of Leadville, Colo.

Monarch district, Colo., began to be large producer of lead, output attaining maximum in 1885.

Destruction by fire of mill and mine buildings of St. Joseph Lead Co., at Bonne Terre, Mo.; replaced immediately by large and improved works.

The Viola mine, at Nicholia, Lemhi county, Idaho, began to be productive.

Tariff on lead continued at 2c. per lb.

1884.

Aspen, Colo., began to produce a considerable quantity of lead-ore.

The Neihart district of Montana began to attain prominence.

Opening of extensive bodies of lead-carbonate ore at Cook's Peak, Grant county, N. M.

First discoveries in the Cœur d'Alene district, Idaho.

Destruction by fire of works of Desloge mine, at Bonne Terre, Mo., and purchase of mine by St. Joseph Lead Co.

1886.

Mines at Aurora, Lawrence county, Mo., began to be developed.

Discovery of Wardner district on the South fork of the Cœur d'Alene river, Idaho.

First production of lead-ore in the Cœur d'Alene district, Idaho.

Mexican lead-ore, especially from Sierra Mojada, began to be imported into the United States in important quantity.

1887.

Opening of Doe Run mine, near Farmington, Mo.

Contest between local and valley smelters in the market for Leadville ore, with advantage in favor of latter, owing to railway discriminations.

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1888.

Attempted corner in the lead market, leading temporarily to high prices, but resulting finally in the failure of Corwith, the chief speculator.

1890.

Establishment of the silver-lead smelting-industry in Mexico, the rapid development of which greatly reduced the supply of Mexican ore available for reduction by American smelters.

Development of disseminated ore at Flat river, Mo.

Tariff on lead continued at 2c. per lb., and lead in ore made dutiable at rate of 1.5c. per lb.

Completion of Mississippi River & Bonne Terre railway and removal by St. Joseph Lead Co. of its smelting-furnaces from Bonne Terre to Herculaneum.

Completion of Northern Pacific and Oregon Railway & Navigation Co.'s tracks into the Cœur d'Alene district.

1891.

Discovery of silver-lead ore at Creede, Colo.

Incorporation of the National Lead Co., this concern succeeding the National Lead Trust, organized a few years previously.

1892.

Development of large bodies of silver-lead ore at Cook's Peak, N. M., and heavy shipments from that point.

The Maid of Erin mine, Leadville, Colo., shipped its last lot of lead-carbonate ore in December, exhausting its great deposit and practically marking the end of the production of this class of ore at Leadville.

Strike of miners in the Cœur d'Alene district on account of reduction in wages.

Invention of the Howard skimmer for handling zinc crust, which was one of the most important of the mechanical improvements in the Parkes process of desilverization. First put into practical use at the works of the Pueblo Smelting & Refining Co., Pueblo, Colo. The Howard press was invented a little later.

1893.

The report of the Herschell committee, closing the Indian mints to the private coinage of rupees, was published June 26, causing a decline in the price of silver from 81c. to 62c. per oz., and contributing to the industrial panic which occurred this year, leading among other things to the suspension of operations in many silver-lead producing districts of the United States.

All of the mines in the Cœur d'Alene closed temporarily on account of low prices for lead and silver.

All of the smelters at Leadville suspended operations in the autumn, only two of them subsequently resuming.

Importations of small amounts of lead-ore from British Columbia.

1894.

Second strike of miners in the Cœur d'Alene.

Tariff on pig-lead reduced to 1c. per lb., and on lead-ore to 0.75c. per lb.

Formation of association of the principal smelters of Colorado to limit prices to be paid for ores. The combination went to pieces early in 1895 and sharp competition was again inaugurated.

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1896.

First patent secured on the Huntington-Heberlein process, a far-reaching and revolutionary improvement in the metallurgy of lead. The process was developed at Pertusola, Italy, by Thomas Huntington, an American citizen, and Ferdinand Heberlein, a German.

Strike of miners at Leadville, Colo., which practically stopped all production during the last six months of the year.

In August of this year the price for lead fell to the lowest point on record in the United States, 2.60c. per lb., New York, having been accepted for several lots. The average for August was 2.73c., and for the year, 2.98c. The lowest price at St. Louis was 2.43c.

1897.

Tariff on pig-lead raised to 2½c. per lb., and on lead in ore to 1.5c. per lb. (Dingley bill.)

The old works of the St. Louis Smelting & Refining Co. at St. Louis, which had been idle for a long time, were again put in operation to smelt ores for southeastern Missouri and the Joplin district. From this time St. Louis increased rapidly in importance as a center of lead production.

1898.

Organization of Empire State-Idaho Mining & Development Co., the beginning of consolidations in the Cœur d'Alene.

The St. Louis Smelting & Refining Co., a constituent of the National Lead Co., acquired property in the disseminated district of Missouri and began its development, leading to a large production of lead in the course of a few years.

1899.

Organization of the American Smelting & Refining Co., which acquired a large number of the silver-lead smelting- and refining-works of the United States. Several of these were promptly dismantled.

Third general strike of miners in the Cœur d'Alene, dynamiting of the Bunker Hill & Sullivan mill, April 29, proclamation of martial law, and final reopening of the mines on a non-union basis.

Entrance of the Guggenheims, under the name of the Federal Lead Co., into the disseminated district of southeastern Missouri.

Strike of smelter workmen in Colorado early in June hindered operations for many weeks.

Great increase in use of lead for electrical purposes (covering cables, etc.).

1900.

Organization of Guggenheim Exploration Co., which acquired, among other property, the capital stock of the Federal Lead Co. and of the Missouri Smelting Co.

1901.

Absorption by the American Smelting & Refining Co. of the smelting interests of M. Guggenheim's Sons, the latter becoming, however, the dominating factor in the amalgamated company.

The American Smelting & Refining Co. assumed control of the lead market, fixing the price both for producers and consumers, and regulating the output by agreement with the large producers and by adjustment of its smelting-charges in connection with small producers. A very large accumulation in the stock of lead on hand occurred this year, which, however, was successfully disposed of in 1902 and 1903.

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1902.

Betts's electrolytic lead-refining process installed at Trail, B. C.; the first electrolytic lead-refinery to be put in practical operation.

Strike of miners at Flat river, southeastern Missouri.

Output of Cœur d'Alene district, Idaho, limited by arrangement between the leading producers and the American Smelting & Refining Co.

The American Smelting & Refining Co. put the marketing of its lead on a contract basis, filling orders for prompt shipment only at a premium of 2.5c. per 100 lb., this being done to induce consumers to cover their requirements ahead and carry the stocks that formerly the smelter often had to carry.

Further steps were taken by the American Smelting & Refining Co. to centralize its smelting-operations, the Philadelphia plant, at Pueblo, Colo., being closed, and the famous old smeltery and refinery at Kansas City being abandoned and soon afterwards dismantled.

1903.

Consolidation of many of the large mines of the Cœur d'Alene by the Federal Mining & Smelting Co.

Organization of the United Lead Co., which secured control of nearly all the manufacturing plants making sheet-lead, pipe and shot, 21 in number, together with a few white-lead works.

Western Mining Co. organized as a subsidiary company of the Guggenheim Exploration Co., acquiring several of the principal lead-producing mines at Leadville, Colo.

In July there was a strike of the smelter-men at the Grant works of the American Smelting & Refining Co., and the plant was closed by the company and abandoned.

1904.

Termination of miners' strike in southeastern Missouri. The labor troubles in this district had been a festering sore for two or three years.

1905.

Adoption of the Huntington-Heberlein process by the American Smelting & Refining Co. It had previously been introduced in Italy, Germany, Spain, Great Britain, New South Wales, Tasmania, Mexico and British Columbia.

Redirection of attention to many of the old mining-districts, including Cerro Gordo, Cal., and Eureka, Nev. Consolidation of Eureka and Richmond companies.

Purchase of the Selby works, at San Francisco, by the American Smelters Securities Co., a sub-company of the American Smelting & Refining Co.

Organization of the United States Smelting, Refining & Mining Co., taking over several independent works, with plans to enter into competition with the American Smelting & Refining Co.

1906.

Reopening of many old silver-lead mining-districts, idle for from 10 to 30 years previous, including Eureka, Nev., and Cerro Gordo, Cal.

The United States Smelting, Refining & Mining Co. erected an electrolytic lead-refinery near Chicago, Ill., this being the first works of this kind in the United States.

The Guggenheim interests practically secured control of the National Lead Co., thus bringing the major part of the lead-consuming industry of the United States into direct affiliation with the American Smelting & Refining Co.

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Physical Factors in the Metallurgical Reduction of Zinc Oxide.

BY WOOLSEY McA. JOHNSON, NEW YORK, N. Y.

(Toronto Meeting, July, 1907.)

INDEPENDENTLY of the recognized chemical reactions involved in the production of metallic zinc, the process is affected by physical conditions in efficiency, and by commercial as well as technical economy. To offer some observations concerning these conditions is the purpose of the present paper.

Among the important elements of this problem is the physical nature of the particles constituting the mass of zinc oxide to be reduced. This may be the result, either of the manner of the original deposition of the mineral, or of the treatment to which it has been subjected in the roasting-furnace. The physical character of the reducing-agent is another important factor, since the carbon of some coals is much more active in reduction than that of others. The degree of fineness, whether of ore or coal, or both, likewise plays a significant part. These factors, together with the physical admixture of foreign substances with the ore, may give rise also to chemical "side-reactions," sometimes hindering or retarding, and sometimes accelerating, the reduction. Finally, thermal conditions, including the relative conductivity of the materials, deserve to be taken into account.

In firing a furnace with retorts charged with a mixture of roasted ore and coal, the first reaction is naturally the boiling-off of the contained water; the next is the distillation of the light hydrocarbons from the coal; the next is the reduction of the iron oxide to protoxide and sub-oxide successively, and then to metallic sponge; and the final reaction is the reduction of the zinc oxide in the ore by the carbon to metallic zinc. This reaction commences at 1,022° to 1,060° C., according to the physical nature and condition of the coal and ore.

Since the retort and the charge are far from being good con-

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ductors of heat, we may assume that there exists in the charge a series of concentric hollow cylinders of gradually decreasing diameter, each hollow cylinder differing in temperature by a few degrees as the diameter grows smaller, the hottest part, of course, being the cylindrical layer next to the retort, and the coldest the core in the center of the charge. This is a convenient assumption for the purpose of illustration. In fact, such hollow cylinders do not exist separately, except as representing zones of temperature.

Now, with rapid firing, it is possible that the charge next to the retort-walls may reach a temperature above that of the reduction of zinc, and, consequently, produce some zinc-vapor, while in the center of the charge, water is being boiled off. Obviously, this would be a bad way to fire the furnace, since the zinc-gas would be thereby so diluted that condensation would become practically impossible. The same evil thing occurs through the distillation of the hydrocarbons when too much light "gassy" coal is used. Diluting gas may similarly be formed from the reduction of the iron oxide when ores very high in iron are treated.

If we had an ideal charge, perfect in heat-conductivity, it might be possible to keep all the contents of the retort at one temperature until the reaction requiring that temperature was complete; then immediately to elevate the temperature until the second reaction was complete, and so on, through the third and fourth reactions. This procedure would require, also, ideal conditions from a chemical standpoint, including a charge of such character that each of the several reactions would be practically complete at a temperature only three or four degrees above that at which it began. Such ideal theoretical conditions are beyond our reach, but may serve as a mental guide for formulating such conditions as are practically the best for any given situation as to ore and coal.

The firing of a zinc-furnace for reducing ores high in iron should be done as follows: The furnace should be heated up to, and kept at, a temperature slightly below the reduction-temperature of zinc, but as near that point as possible. This, in the course of an hour and a half or two hours, will bring all the contents of the retorts to a uniform temperature of slightly below 1,020° C. Below this temperature the reduction of the

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iron oxide is largely completed. After this has been accomplished, the furnace should be fired for the reduction of zinc oxide.

If the reduction of the iron oxide and the reduction of the zinc oxide proceed simultaneously, the evolution of so much carbon monoxide and dioxide from the reduction of the iron oxide would sweep the zinc-vapor through the condenser and burn it at the mouth. This is exactly what happens with too-rapid firing. The bright or so-called "angry" flame, burning the zinc with brilliant white fumes, while yielding practically no condensed metal, is the practical fact of which the above paragraphs are the explanation.

Through both special experiments and practical experience in zinc-smelting, I have found that the iron oxide has different degrees of reductivity, varying even more than that of zinc-ores, according to the way they were originally deposited geologically and the way they have been roasted. This result corroborates the observations of Sir Lowthian Bell. A hard, dense zinc-ore that is hard to roast, and gives a product not very porous, is naturally more difficult to reduce than one of a porous nature, which has been roasted at a low temperature. If, as sometimes happens, a zinc-ore is roasted at such a temperature as to form an incipient slag—for example, an iron silicate—and also compounds of its different constituents, such as zinc ferrate, zinc silicate, zinc aluminate, and lead silicate, the reduction of such an ore, so roasted, is retarded, its reductivity being decreased, because the compounds of zinc oxide are more difficult to break up than the oxide itself. Moreover, the particles are shrunk and cemented more closely together by the heat, just as a fire-brick, under the action of high heat, contracts and hardens; and their reductivity is thus decreased for physical reasons.

Thus it may be said that there are two kinds of work to be done in the reduction of zinc oxide. One is chemical work; the tearing of the atoms of zinc and oxygen away from each other against the force of affinity. The other is the work of tearing the molecules of zinc oxide from one another. This is physical work. Any agent that changes the aggregation of the molecules so as to make it denser increases the physical work to be done in reduction.

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The formation of iron silicate in roasting is especially bad; for it is not easily reducible, and is itself a slag already formed, which will pick up other slag-materials, and, collecting in the bottom of the charge, quickly cut a hole through the retort. For this reason, several zinc-plants in Kansas charge carbonates "green" into the retort, even though their bulk is greater.

Metallic iron as an iron-sponge does not, in my opinion, have very deleterious effects, provided sufficient fine clean coal is present. In the first place, the iron seems to draw up in globules in the pores of the charge by capillary attraction, and to be held there as water is held by a sponge. Not only is its melting-point high, but it has no great chances to form alloys with other metals.

The iron sulphide, on the contrary, is really the most corrosive part of a charge, whether present in the charge as pyrites in the original coal or pyrites not roasted in the ore, or formed by the iron sponge acting on the zinc sulphide. At from 1,100° to 1,200° C. iron sulphide will dissolve fire-clay practically as hot water dissolves sugar. Several times, in my small electric crucible-furnace, I have added pieces of a retort to molten iron sulphide, and observed a considerable evolution of sulphur dioxide, suggesting that an iron silicate formed. This product analyzes 3 or 4 per cent. S and is partially a sulpho-silicate.

Moreover, molten iron sulphide is extremely mobile and will penetrate the pores of a retort, where it forms, under the oxidizing flames of the combustion-gases, an iron slag, and cuts a hole through the retort. Then the metallic iron in the charge is oxidized and furnishes enough iron oxide for the formation of more slag. In the course of four or five days, if slagging has happened first on a retort in the top row, all the retorts vertically under the unfortunate one will be cut out and spoiled by the slag dropping down from it. The whole trouble is probably caused by an accidental and local excess of iron sulphide in one portion of the charge, due to poor roasting or poor coal, or both, and an uneven "mix."

A phenomenon of zinc-smelting which puzzled me for a long time is what is known as the "setting" of the furnace. When a furnace is fired up so rapidly at first that a great deal of zinc cannot be condensed but burns at the mouth of the

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condenser, the furnace-man will often suddenly reduce the temperature, to stop this loss of zinc. If this change be too marked, the charge in the furnace becomes "set," and it is impossible to get the zinc out, even though the temperature be raised to a point much above that ordinarily employed. The reason is, that a thick heavy slag is thus formed, and, once formed, contracts on cooling, and encircles the particles of ore and coal. If the temperature had been brought up slowly, the slag would never have been formed, or would have been formed only at the end of the shift, when the charge was "dry" of zinc.

This suggests another point. While a thick slag prevents reduction, the formation of a very liquid slag facilitates it, probably by bringing the reacting substances into igneous solution, so that they can act on each other. In American zinc-practice, it is deemed inadvisable to form slag at any time, if it be possible to prevent it, but in Europe, occasionally, the residues are tapped from the retorts as a liquid slag.

This brings us to the essential difference between European and American zinc-furnace practice. By a sort of natural evolution, the practice of each country has tended along the lines suited to the commercial environment. In Europe, coal is expensive and labor cheap. In the United States, gas and coal are very cheap, labor is dear, and our workmen have not had the training in zinc-smelting of those in Germany and Belgium, where this calling has often been followed in the same family for three or four generations.

Consequently, the practices are radically different. The German metallurgist charges a small amount of coal, with a roasted zinc-ore averaging something under 50 per cent. of zinc, into muffles holding from 80 to 100 lb. of ore. This is fired in regenerative furnaces with great care. At the end of the shift, the residues are carefully scratched up by hand, sometimes without taking the condenser down, and the retorts are sometimes charged again through the mouth of the condenser. A great deal of labor is always used per retort.

In the United States, the retorts are cylindrical and are charged very rapidly with a material high (in the case of Joplin ores over 70 per cent.) in zinc. The furnaces are fired very extravagantly, usually with natural gas. At the end of the shift, the residues are blown out by steam or water. In

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Kansas, the aim is always to make a charge that contains so much coal and so little iron sulphide as to be non-corrosive, and leave "dry" residue—one that contains a minimum of slag. Very little scratching or cleaning of the retorts by hand-rabbling is done in this country. It will be seen that the practice of Europe and of America is different in these respects, by reason of totally unlike conditions in the two regions.

In the treating of ferruginous zinc-ores from Leadville and elsewhere, American practice has been simply a refinement and a carrying-out to the extreme of the method pursued with Joplin ores. The attempt is always to make a non-corrosive charge which, by reason of its large amount of fixed carbon, will also have a maximum reductivity at as low a temperature as possible. The criterion of this charge, for low-grade ores, is recognized by the practical zinc-smelter in the glowing and brightening-up of the residues when the condensers are "knocked down" in the morning, at the end of the shift. If the residues then glow, it is, of course, a sign of the presence of active particles of carbon, to be attacked by the oxygen of the air which rushes into the retort. Consequently, the charge, at the end of the shift, still possessed considerable reductivity, and it was not necessary to use intense heat to reduce and distill the zinc. Did the charge contain no remaining active carbon it would not glow. The zinc-smelters have almost a superstition about this glowing or "looking red" in the retort; and it is convincing to see that it has good chemical reasons.

Another fact about zinc-reduction which I have never seen noticed, is that carbon-deposition occurs around the iron oxide just as it does in the shaft of an iron blast-furnace, and that this deposited carbon is very active in reducing the particles of zinc oxide.

The reaction of the iron-sponge on both zinc oxide and zinc sulphide increases, of course, the amount of zinc distilled. These reactions are both completed below 1,200° C. The iron is formed as an iron-sponge through the reduction of the iron oxide by the carbon of the charge. For this reason, magnetic zinc-blende, the so-called "marmatite," is very active.

Another characteristic of the reduction of zinc oxide is the distillation of the heavy tars in the coal. These heavy tars are distilled at about the temperature at which zinc is reduced, and

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are retained in the condenser, where they form a solid mass, and in time choke the condenser. I have never seen this reason given before; but it is undoubtedly the chief reason why accretions are slowly formed on the sides of the condensers, which have to be cleaned out by the "connie boy."

The coal for a zinc-charge should have maximum heat-conductivity, "reductivity" (*i. e.*, active energy in reduction), and percentage of fixed carbon per cubic foot; and minimum proportion of hydrocarbons and of ash (the ash containing, moreover, a maximum of aluminum silicate). Finally, it should be low in price.

To combine all these good qualities in one kind of coal is impossible, and hence it is advisable to use three or four different kinds, one serving one purpose, and another another. For instance, petroleum coke, the residue from the distillation of oil, contains from 94 to 96 per cent. of carbon, and less than 1 per cent. of ash; but it carries 3 or 4 per cent. of heavy hydrocarbons. This would be an ideal reducing-agent for zinc, did not its hydrocarbons choke up the condenser; and therefore it should not be used for more than one-sixth of the total fuel.

"Dead coal," from the "strip-pits" of Missouri and Kansas, has carbon of good activity. The ash is also pretty infusible, because the sulphur has been extracted through weathering of the coal. It yields, however, a very high amount of gas, and therefore too much of it should not be used. Moreover, it is always well to use several kinds of coal, so that accidental poor shipments of one kind will be "averaged out."

All things considered, probably the best reducer for a zinc-charge is anthracite slack, having less than 0.6 per cent. sulphur and an ash which is nearly an aluminum silicate. It is, however, usually too expensive for use except in a few localities.

For maximum reductivity, neither the zinc-ore nor the reducing-material should be too fine. With coal, and ore finer than 60-mesh, there will be channeling of the gases of reduction, and the very efficient reducing-action of hydrocarbon vapors and carbon monoxide will be lessened. Besides, there is a "back-pressure" of these gases, which retards reduction.

The ideal of American zinc-practice should be to produce a charge of maximum reductivity with an infusible residue. If

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this were attained, it might be possible to use much larger retorts, or possibly a gas-fired continuous furnace.

In this paper the word "reductivity" has been applied to the charge, the reducing-agent, and the ore, while possibly a stricter phraseology should have been used.

In conclusion, I want to express my appreciation of Dr. Percy's early work on zinc. Having repeated and extended all his experiments, I feel bound to testify that they reached the heart of the subject, and established principles not superseded by later progress. This is not true of all such pioneer work, however creditable it may have been in its day; and when it is true, it deserves to be publicly declared.

I wish, also, to acknowledge my indebtedness for valuable suggestions to Messrs. C. A. H. de Saulles, John Nett, Walter J. Chapman, and many others.

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Geology and Mining of the Tin-Deposits of Cape Prince of Wales, Alaska.*

BY ALBERT HILL FAY, B.S., E.M., NEW YORK, N. Y.

(Toronto Meeting, July, 1907.)

In giving a sketch of the geology and mining of the tin-deposits of Cape Prince of Wales, a short description of the geographic and climatic conditions may be of special interest on account of this being a part of the world of which very little is known, even by the reading public. As shown in Fig. 1, the location of the Cape is lat. N. 65° 35′ and long. W. 168°. It is bounded on the north by the Arctic ocean; on the west by Bering strait and on the south by Bering sea. It is the westernmost point of the mainland of North America, and to the west, across the strait, one can see East Cape, Siberia, which consists of a rugged, steep coast of granite cliffs.

The climatic conditions are of interest because of the part they have had in the surface geology, and also because they affect mining from an economic standpoint. In connection with the climate and geography, it does not seem out of place here to quote a few lines from the prologue of *The Wandering Jew*, written by Eugene Sue, under the caption "The Land's End of Two Worlds."

"The Arctic Ocean encircles with a belt of eternal ice, the desert confines of Siberia and North America—the uttermost limits of the Old and New World, separated by the narrow channel known as Bering's Straits.

"To the north, this desert is bounded by a ragged coast, bristling with huge black rocks. At the base of this Titanic mass lies enchained the petrified ocean (Bering sea) whose spell-bound waves appear fixed as vast ranges of ice mountains, their blue peaks fading away in the far-off frost-smoke, or snow-vapor.

"Between the twin peaks of East Cape, the termination of Siberia, the sullen sea is seen to drive tall icebergs across a stream of dead green. There lies Bering's Strait. Opposite, and towering over the channel, rise the granite masses of Cape Prince of Wales, the headland of North America.

"These lonely latitudes do not belong to the habitable world."

* Submitted, also, with the consent of the Council of the Institute, as a thesis in partial fulfillment of the requirements for the degree of Master of Arts, to the Faculty of Pure Science, Columbia University, New York, N. Y.

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Such is Sue's description of this barren and desolate country, and it is so true, that any navigator upon the high seas, who had read the prologue, could not do otherwise than recognize the place though he had never previously been here.

During eight months in the year no boats can reach Cape Prince of Wales. In the summer, the United States mail arrives twice a month; and in the winter there is a weekly mail

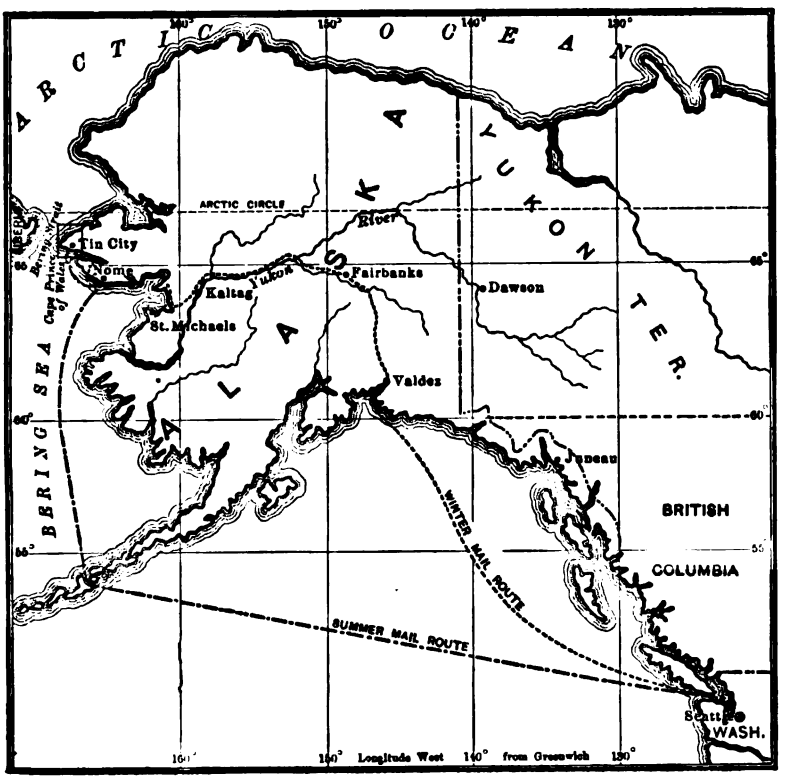

FIG. 1.—SKETCH-MAP OF ALASKA.

service from Nome. The winter mail is carried on horse-sleds from Valdez to Fairbanks over the trail shown in Fig. 1; thence down the Yukon river to Kaltag; across the portage to Unalakleet; and along the coast line to Nome and Tin City. From Fairbanks to Tin City the mail is carried by dog-sled. The total distance from Valdez (the winter sea-port) is a little more than 1,400 miles. Mail from the United States will reach the Cape in from 60 to 80 days. There is a telephone line to

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TABLE I.—*Daily Temperatures at Tin City, Alaska*, 1906–1907.

	DAY OF MONTH.																														Average.
	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22	23	24	25	26	27	28	29	30	31	
1906 AUGUST.									55	48	52	41	37	44	42	46	44	54	60	48 s	44 s	42 s	54 s	50 s	44 f	40 s	32 s	36 s	37 f	39 f	36 f	44.7
SEPTEMBER.	38 f	40 f	36 f	36 f	36 s	36 s	34 s	40 f	48 f	44 f	38 f	40 f	40 f	45 f	38 f	40 f	40 s	38 s	40 s	39 s	40 s	32 s	34 s	34 s	33 f	34 f	36 f	37 f	34 f	30 s		37.6
OCTOBER.	32 s	30 s	33 f	35 f	37 f	35 f	31 f	30 s	30 s	26 s	26 s	27 s	28 s	25 s	28 f	30 f	30 f	34 f	36 s	35 s	34 f	28 f	30 f	32 f	30 f	28 f	30 s	28 s	30 s	34 s	26 s	30.6
NOVEMBER.	24 s	25 s	18 s	20 f	24 s	26 f	30 f	18 f	16 f	0 f	6 f	38 f	16 f	16 f	12 f	8 f	10 f	3 s	6 s	3 s	-4 s	0 s	0 f	-2 f	1 f	-3 f	-2 f	4 f	8 s	10 f		10.8
DECEMBER.	-10 f	0 f	8 f	0 f	2 f	8 f	7 s	2 f	3 f	0 f	-10 f	-9 f	19 f	4 f	20 f	28 f	1 s	-2 s	-2 s	0 s	1 f	4 f	5 s	14 f	24 f	18 f	-5 f	14 s	18 s	18 f	26 s	6.6
1907 JANUARY.	25 s	28 s	8 f	24 s	7 f	28 s	20 s	15 s	-28 f	10 f	24 s	28 s	26 s	5 f	4 s	-12 f	-12 f	-24 f	-15 f	8 s	-6 f	-12 f	12 s	18 f	12 f	12 f	6 f	10 f	6 f	0 f	4 f	7.5
FEBRUARY.	10 f	2 f	0 f	-16 s	-26 s	-22 s	-16 s	-22 s	-23 s	-23 s	-34 s	-42 s	-37 s	-37 s	-40 s	-32 f	-31 f	-26 f	-15 f	-36 f	-36 f	-10 s	0 s									-22.2

f—Fair Weather. s—Stormy Weather.

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Nome, where it connects with the United States military telegraph line to Seattle.

Climate.

The summers are comparatively cool, 60° F. being the highest temperature, while 45° is about the average from June 15 to Sept. 15. At Cape mountain a very large part of the time (at least one-half) there are heavy fogs and drizzling rains, which make it bad for all outside work. The wind blows very hard at times, adding a great deal to the inclemency of the weather. Freezing temperatures, with snow-storms, usually set in about Sept. 15. Navigation for small boats from Nome is uncertain after Oct. 20, although in 1906 they continued to run until Nov. 10. By Dec. 1 the sea is usually sufficiently frozen to permit traveling by dog-sled (Fig. 8) on the ice near the beach. The winters are noted for very severe blizzards of snow-ice and high winds. Yet the temperature does not drop so low as in more inland places. The lowest temperature for the winter of 1905–6 was —45° F. During these blizzards, which often last ten days at a time, it is almost impossible for man or beast to endure the icy blasts as they hurl themselves down upon him from the polar regions. The changes in temperature are very sudden, as shown in Table I., which I compiled from daily observations from Aug. 9, 1906, to Feb. 23, 1907. In 24 hr. during Jan. 8 and 9, 1907, there was a drop of 43°, and the maximum range for the period observed was 102°. February was a cold month, with a continuous blizzard lasting about two weeks.

Table II., the weather record[1] covering a period of years at the Government station at St. Michaels, is here given to show something of conditions at that place. St. Michaels is about 225 miles SE. of the Cape, and is the place nearest to Tin City where records have been kept for any length of time.

[1] Gravel and Placer Mining in Alaska, by C. W. Purington, *Bulletin* No. 263, *U. S. Geological Survey*, pp. 48, 49 (1905).

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TABLE II.—*Rainfall and Temperature at St. Michaels.*

Month.	Rainfall. Average 7 yrs. 6 mos.	Rainy Days.	Temperature. Average 11 yrs.
	Inches.		°F.
Jan.	0.9	8.1	7.4
Feb.	0.2	5.5	-2.3
Mar.	0.5	7.4	8.9
Apr.	0.4	7.8	19.9
May	1.3	9.1	33.1
June	1.5	10.4	46.3
July	2.5	13.6	53.6
Aug.	3.3	16.7	51.9
Sept.	4.0	18.5	43.9
Oct.	1.7	11.4	30.5
Nov.	1.2	11.4	15.6
Dec.	0.8	6.9	4.8
Total.	18.3	126.8	Average 26.1

During seven months of the winter the climatic conditions permit little, if any, outside work, although freighting, such as hauling ore, could be performed during about half of the winter season. Underground work can be carried on without difficulty during the entire winter, but it is absolutely necessary to get all supplies in before Oct. 1. The ground is frozen very deep, and some prospect-shafts and tunnels from 80 to 100 ft. below the surface are still in the frost-zone. No well-water has been found; and, for domestic purposes, the supply comes from thawing the snow as needed. Water can be obtained near the beach at a depth of 12 ft., but it is too salt for any purpose other than milling. A supply of water for mill use during the summer can be obtained from the melting snows on the mountains and needs no pumping.

The operation of any concentrating-plant in which water takes an important part can be carried on only during four months in the summer, since cost of fuel will be prohibitive in winter.

VEGETATION.

On account of the long winters and very short, cool summers, it is surprising to find such a variety of plants. There are no trees, nor even shrubs. The vegetation consists almost exclusively of moss and small flowering plants, of which the forget-me-not is abundant. I have seen during the short sum-

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mer 30 or 40 varieties of these delicate flowers, nearly all having bright colors and most fragrant odors. It is not uncommon to find these tender plants growing alongside of deep snow on the slopes of the hills. There are several species of moss, which furnish food for the reindeer, and though grass is scarce, enough plants grow to give the low, flat places a bright green appearance during August. The sunshine in summer lasts from 18 to 23 hr. a day, and under its influence plants come out very quickly after the snow leaves the ground bare, and mature quite rapidly.

Geology.

Fig. 2, of Cape mountain, gives a good idea of the topography of this portion of Seward peninsula. When the geology of

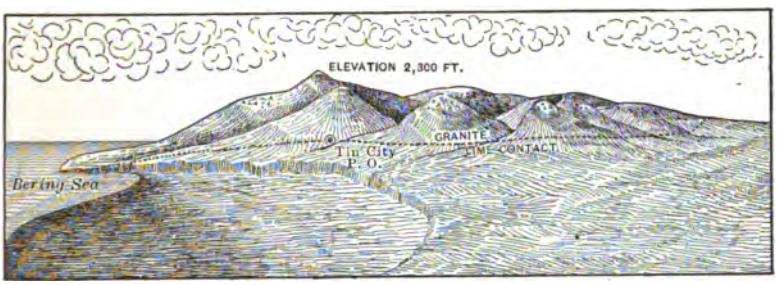

Fig. 2.—Cape Mountain, from the East, Showing Topography.

the country covered by the map, Fig. 3, is interpreted we are led to the belief in a very marked eruptive period, followed by a long time-interval, during which erosion has played an important part. The western end of the mountain, next to the Cape, consists of a granite knob 2,300 ft. high, known as Cape mountain, which rises so abruptly from Bering sea that there is not enough strand line for even a foot-path. This is indicated in Fig. 8. To the north the mountain slopes gently to the Arctic ocean, where there is quite a broad margin of low ground, formerly sand-spits, but now covered with tundra. These sand-spits are still forming, and inclose a series of lagoons along the coast, the largest one being known as Lopp lagoon. East of Tin City there is a broad tundra-covered plateau extending to the York mountains, with an elevation of from 300 to 400 ft. above sea-level. This plateau is very

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much dissected by short valleys extending north and south, which, during the summer, are water-ways for the small streams. On the western side of this plateau, and against the

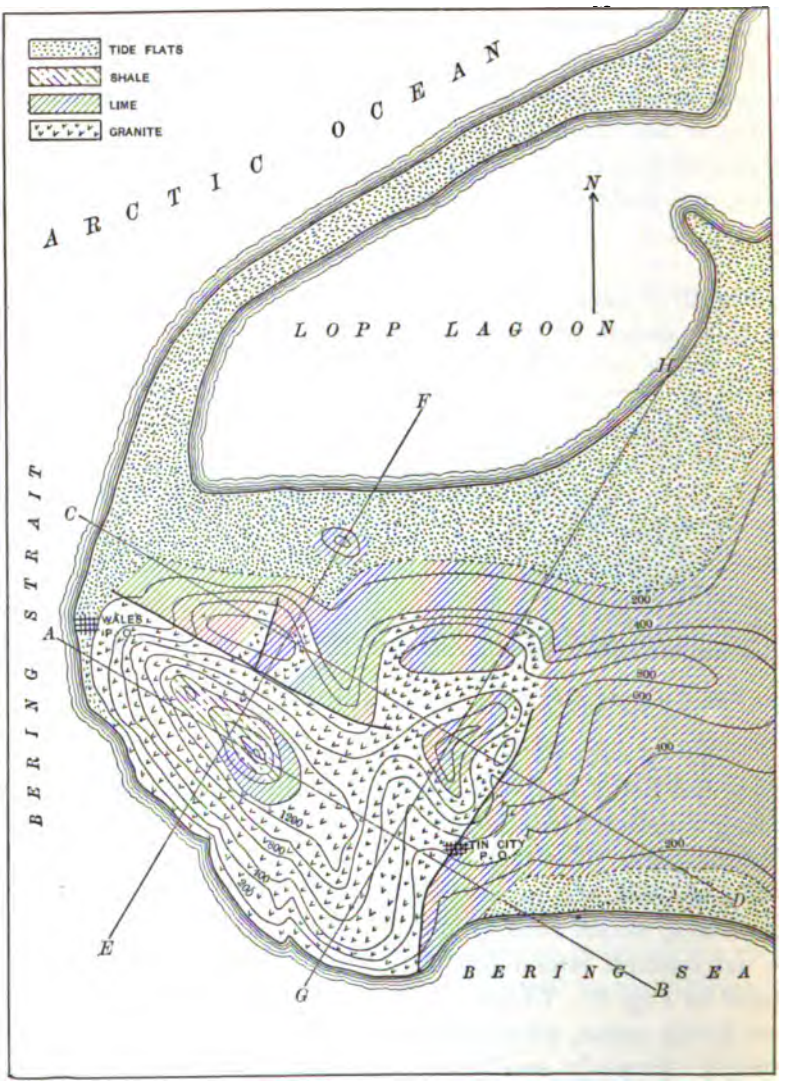

FIG. 3.—GEOLOGICAL MAP OF CAPE MOUNTAIN, CAPE PRINCE OF WALES, ALASKA. (Scale: ¼ in. = 1 mile.)

granite mountain-mass, are three limestone terraces, which have the appearance of ancient beach-lines. The lower one is now very near the present beach, and no doubt the present

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narrow margin along the coast is, in places, of very recent formation. The upper terraces I am inclined to consider as the remnants of a monocline—or the west limb of an anticline, in which the erosion has progressed westward from a N–S. axis of uplift. Six or eight miles east of these terraces, which are near the granite-lime contact, the strata dip in the opposite direction, and thus indicate the presence of the two limbs of an anticlinal fold.

The surface of the more elevated and mountainous portions shows the effect of frost in a very marked degree. The freezing and thawing process has gone on so long, and the outcropping rocks have been broken up to such an extent, that practically all the surface is covered with loose fragments to a depth of from 5 to 20 ft. On the steep mountain-slopes this *débris* creeps so much that when float-ore is found, one can hardly even guess its point of origin. Except for some of the granite monoliths and an occasional limestone cliff, outcrops are scarce, making prospecting very difficult.

There are a large number of these granite monoliths from 20 to 75 ft. high and of grotesque shapes, standing in clusters, as shown in Fig. 9. One of these columns, Fig. 10, has the profile of a man's head and is named the "Wandering Jew."

The granite rocks, Fig. 9, are exceptionally good examples of wind-erosion. These rocks stand in approximately a N–S. line, with the north at the left of the photograph. By close observation, it will be noticed that the left side of these rocks is very much rounded and cut away, while the right or south side is very angular. The prevailing wind at this place is from the north, and it usually blows with a high velocity. At times the force is so great that it is almost impossible for a man to keep on his feet. In summer the wind carries more or less sand, which acts as a sand-blast on these giant rocks, and in winter vast quantities of snow and ice are carried by the wind and assist, though to a lesser extent, in the destruction of the monoliths. The view of the Wandering Jew, Fig. 10, does not show the work of the wind, because this lone rock stands on the side of a hill where it is very much protected.

The granite is somewhat coarse and light colored. The feldspar-crystals vary in size from a small fraction of an inch to 2 in. long. The hornblende-crystals are few and small, and

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biotite is the prevailing dark mineral. The main body of granite has the appearance of a boss, on the top of which is a lime and shale cap. From this main boss extend intrusions into the lime between the bedding-planes, and form sills which are nearly horizontal. There are also vertical dikes, from a few inches to 30 ft. thick, radiating from the central core and extending into the lime. These dikes appear to have the same texture as the granite along the main contact, and are probably contemporaneous with it. At one place I found a dike of rhyolite-porphyry cut through the coarse granite. The contact between the two masses is very sharp and showed no decomposition of either. At the contact the rhyolite and granite were actually fused together.

There were at least three periods of disturbance after the deposition of the limestone: 1. Eruption of the main granite-mass. 2. Intrusion of rhyolite-dikes. 3. Intrusion of basalt-dikes.

After the limestone had been deposited came the great uplift, which formed the mountain, and tilted the limestone-beds, shown in Figs. 4, 5, 6 and 7. Then came a period of rest and the whole mass solidified. Following this was very likely a long time-interval, during which erosion played an important part. Then came the intrusion of rhyolite-dikes, as mentioned above. I also found one basalt-dike which cut through both the granite and lime, at the contact of the latter. No place was found where the basalt- and rhyolite-dikes cut each other; but on account of the acidity of the rhyolite it is probably the older of the two. The basic rocks have a lower freezing-point. It is not at all likely that the acid and basic dikes were intruded at the same time, for the two observed were only a few yards apart.

The limestone still retains its bedding-planes, and is generally of fine grain and bluish color. In places near the contact with the granite it has become crystalline, while at others it is more siliceous, with stringers of quartz between the bedding-planes; a large amount of the lime at the contact near the beach, as well as at the contact near the top of Cape mountain, shown in Fig. 11, has altered to wollastonite. I found neither garnet nor fluorite in this contact-zone.

There are four places where there is still some limestone on

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top of the granite, as shown in Fig. 3. Toward the top of Cape mountain, as well as near the Indian village, the limestone becomes more shaly, and even schistose in character.

I found three fossils in the lime, one near the Indian village (Wales P. O.) and one three miles east. Another was found

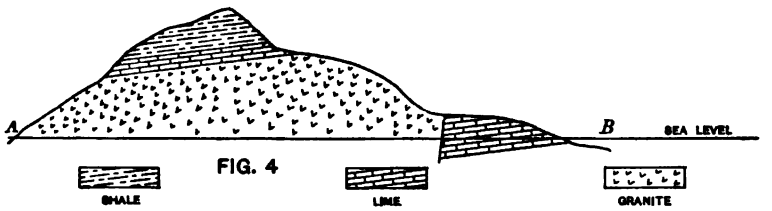

FIG. 4

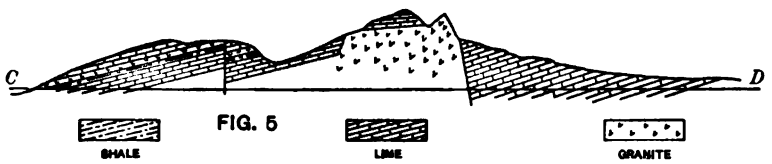

FIG. 5

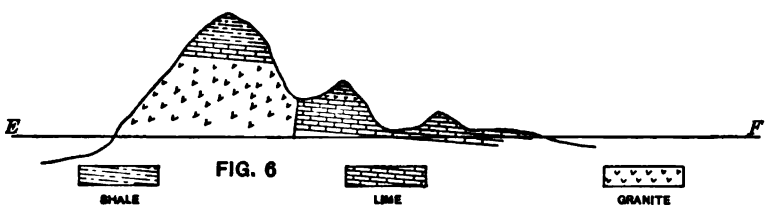

FIG. 6

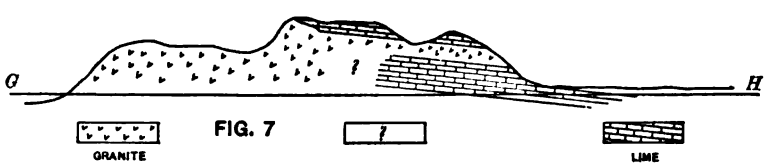

FIG. 7

FIGS. 4, 5, 6 AND 7.—GEOLOGICAL SECTIONS, CAPE MOUNTAIN.

on the beach about two miles south of the last one above mentioned. All three were alike and silicified, but no report has yet been made. Mr. A. J. Collier has reported that the fossils from this section belong to the Carboniferous age.[3]

[3] History of Explorations and Surveys in Geography and Geology of Alaska, by A. H. Brooks, *Professional Paper* No. 45, *United States Geological Survey*, pp. 206, 217, 224 (1906).

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As shown in Figs. 4, 5, 6 and 7, the limestone dips to the west at an angle of 5° to 15°, with a strike of N. 10° E.

Near the beach the contact-line between the limestone and granite is vertical, and about two miles farther north it dips east 80° (Fig. 5). One-half mile farther to the north it dips 60° W. The slickensides show that some movement has taken place along this line.

In addition to the main contact, there are occasional bodies of limestone which have been caught in the molten granite, and are now held as inclusions. It is alongside one of these that the best tin-ore has been found. This particular body of limestone is about 20 ft. thick.

Tin-Deposits.

At a number of places on Cape mountain a little float tin-ore has been discovered, but not enough to be of importance. The lode-deposits are also small, at least as far as present developments show. Much of the granite carries traces of tin, and occasionally small stringers of cassiterite exist along very fine fractures in it. In one case a stringer from 1 to 2 in. thick was found which was nearly vertical. This stringer passed through the granite, and also along the bedding-planes of the limestone, showing the tin to be later in origin than the limestone.

The cassiterite found in the limestone was crystalline and exhibited good faces. Associated with it was quartz and limonite. The stringer of cassiterite in the granite was more massive and had the appearance of a local enrichment. There was no marked line of contact; the center was almost pure cassiterite, and passing from this to the granite the tin oxide seemed to give place to the biotite and feldspar until the normal granite was reached on either side. At one point of contact there is an iron selvage varying from a few inches to 4 ft. thick. The iron oxide (limonite) is quite pure and contained no tin. The tin at this place seems to be in the granite.

Along this granite-limestone contact there is an excellent example of the formation of tourmaline. All the necessary conditions are provided. The granite above the limestone lies almost horizontal, and must have provided, when in the molten and viscous state, practically a reverberatory furnace. This overlying mass, by reason of its weight, would retain emanating gases under pressure. All the constituents of tourmaline are

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FIG. 8.—BERING SEA ICE, EXTENDING DIRECT TO THE BASE OF THE GRANITE CLIFF, WITH MAN AND DOG-TEAM IN THE FOREGROUND.

FIG. 9.—MONOLITHS OF WEATHERED GRANITE ROCKS, FROM 25 TO 75 FT. HIGH.

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FIG. 10.—THE "WANDERING JEW," ABOUT 40 FT. HIGH.

FIG. 11.—LIMESTONE BEDDING NEAR TOP OF CAPE MOUNTAIN. THE WHITE STREAKS, MARKED "W," ARE WOLLASTONITE.

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at hand: lime in the limestone; iron, magnesium, aluminum, sodium, potassium, boron and silicon in the granite. Boric fluoride is very readily absorbed by water (700 volumes of BF_3 to 1 of H_2O), and in this way it may have been supplied by the crystallizing granite. The tourmaline has formed on the lower side of the granite and exhibits well-developed crystals in quantity. As depth in the granite is gained, the amount of tourmaline diminishes. Below the tourmaline-granite, and resting immediately on the limestone, is a quantity of filling consisting largely of limonite and free quartz. This iron has undoubtedly been precipitated here as a sulphide, and is now the product of oxidation. This change has been wrought when climatic conditions were milder than now, for it is all in the frozen zone.

The presence of an occasional stringer of cassiterite in the granite, now in place, would indicate that the float tin-ore on the surface is simply a residual product of decomposition of the granite. It might be used as proof that there are still rich bodies of ore here; but it is more reasonable to regard the float as representing stringers and veinlets that have existed in bygone ages. Erosion has been great, and it is not an easy matter to say to what extent these deposits have been carried away. Placer tin-ore is found in many places on the Seward peninsula, and no doubt all has the same origin.

On the whole, not enough work has been done to obtain very much information concerning the size and continuance of the lode-deposits. All that can be said now is that they are not large enough to pay to work. Some of these stringers may improve as depth is gained, but that remains to be seen.

Sampling and Assaying.

The assay was usually made on 100 g. of pulp of 60-mesh size obtained from the sample, which was taken to represent as nearly as possible an average value. This pulp was carefully panned until most of the gangue had been washed off, leaving a concentrate of about 10 g. containing cassiterite, with small amounts of pyrite, iron oxide and tourmaline, together with some other gangue materials clinging to them. This concentrate was then digested in nitro-hydrochloric acid for an hour, or longer, until all the soluble parts were dissolved,

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leaving most of the tin oxide reasonably clean. The residue, containing some quartz and tourmaline, was then thoroughly washed, dried and weighed. A fire-assay on a number of these pannings gave an average of 70 per cent. of metallic tin, while pure cassiterite contains 78 per cent.

By practice and careful work in the panning and acid-treatment, uniform results can be obtained. I made 300 tests on various samples in this manner and considered my final concentrate as 70 per cent. of metallic tin. Though not absolutely correct, this method of assay is sufficiently accurate for testing country-rock, float-material, grab-samples from ore-dumps, and prospectors' samples. Moreover, when at a distance from a well-equipped laboratory, the best use has to be made of the tools and supplies that are available.

The charge for the fire-assay was made up as follows: 10 g. of concentrates from ore was thoroughly mixed with 40 g. of KCN (98 per cent.); 5 g. of KCN was placed in the bottom of the crucible, the charge then added, and covered with 5 g. of KCN. The time of fusion in gasolene furnace, from 15 to 25 min., gave a metallic tin button ready to be weighed. No analysis of the tin button was made to determine its purity.

Of 36 samples of granite, 13 contained a trace of tin, while 23 showed none. Those that had any tin whatever were the ones nearest the contact, or, in other words, in the outer shell of the granite. Phillips and Louis, in describing a mine in New South Wales, report that "the tin-bearing portion of the granite is practically confined to a belt that forms the outer crust of the boss."[3] The portion of the vein-matter which could be taken as ore for treatment has a very low tin-content.

Mining.

Down to the present time very little mining has been done, but there has been considerable prospecting by a half dozen concerns. A large part of this prospecting has been in the shape of open cuts and costeaning trenches, in the endeavor to find the ledge from which the float-ore has come. The surface of the ground consists of so much loose earth and boulders, that it is very difficult to obtain good results from open work. During the short summers, the ground will thaw only to a

[3] *Treatise on Ore-Deposits*, Phillips and Louis, p. 668 (1896).

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depth of from 3 to 4 ft.; and, below this, the digging is very hard. If the open cut is on level ground, the melting ice and snow will flood it with water, so that it is necessary either to abandon the work, or to put in a pump, before bed-rock can be reached.

Prospecting has been carried on continuously since 1902, the largest operations being those of the Bartels Tin Mining Co. The mining-work consisted of a large number of open cuts from 10 to 100 ft. long, many of which, however, did not reach bed-rock; a number of short tunnels, and two shafts, one 80 ft. and one 40 ft. deep. In two or three places some cassiterite was found *in situ*. The surface-improvements consist of two bunk-houses, two warehouses, a barn, a blacksmith's shop and a power-house for electric drills. A Merrill 3-stamp mill was installed two years ago, comprising one 25-h.p. Model gas-engine, a Blake crusher, a hydraulic classifier, and two Wilfley tables. The plant, which has a capacity of about one ton per hour, was operated during the summer of 1906, and a small shipment of concentrates was obtained.

The next largest company operating is the U. S. Alaska Tin Mining Co., owning a number of claims on the north slope of Cape mountain. The underground work consists of one small and shallow shaft, and a tunnel about 250 ft. long. A 10-stamp mill has been installed.

During the summer of 1906, Risley and Arrowsmith did some prospecting by the use of the calyx core-drills. Several holes were bored to a depth of from 50 to 100 ft. through lime to the granite-contact, but I do not know what results were obtained.

There are several minor prospect-diggings, but they do not seem to be important or prominent.

Economic Conditions.

Table III. gives the "quantity" prices that the mining companies have to pay for supplies if purchased at Nome, also the corresponding prices at Seattle.

The cost of labor during summer and winter is $5 per day, plus board and lodging. At Nome, the rate in winter is $4 and in summer $5 per day, including board.

The outlook for the tin-mining industry at Cape Prince of

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TABLE III.—*Prices of Supplies at Nome and Seattle.*

Commodity.	Price at Seattle.	Price at Nome.	Freight from Nome to Tin City.	Freight from Seattle to Tin City.
Lumber	$13 per M.	$40 per M.	$22.50 per M.	$20 per ton (40 cu. ft.) by sailing-vessels. This makes the freight on hay about $30 per ton and lumber $25 per M.
Coal	$5.90 per ton.	$18 per ton.	$15 per ton.	
Flour (best)	$60 per ton.	$80 per ton.	$15 per ton.	
Fresh beef	.....	15 to 20c. per lb.	¼c. per lb.	
Eggs (30 doz. to case)	$9 per case.	$12 to $14 per case..	$1 per case.	
Coal oil (10-gal. case)	$1.55 per case.	$3.50 to $4.25	75c. per case.	
Hay	$23 per ton.	$47.50 per ton.	$22.50 per ton.	
Oats	$30 per ton.	$52.50 per ton.	$15 per ton.	
Bran	$18 per ton.	$40 per ton.	$22.50 per ton.	
Hams and bacon	.....	19 to 25c. per lb.	¼c. per lb.	

Wales does not seem to be encouraging. A number of shipments of tin-ore (placer) have been made from Seward Peninsula, but all were small, and in every case the amount realized on the shipment was far less than the cost of production. A large number of companies have been organized for the exploitation of the stream-tin deposits, of which many have gone out of business, only to be followed by others trying to work the same ground. So far as I know, the deposits are small and of low value. This condition, and the short working-season in summer, the high price of supplies, freight and labor, combine to retard the progress of any concern that has the production of tin-ore as its object. Similar conditions apply also to all the "quartz" tin-mining, by which term the miners mean tin-ore *in situ*. Not only is the ore low grade, but the quantity is a very uncertain factor. With a large quantity of ore, it might pay to work values of 2 per cent. of metallic tin, in spite of the high cost of everything that enters into its production. Mr. Hess, in speaking of the tin-deposits of the Cape, says:[4] "It seems safe to figure that under present conditions nothing less than 2.5 per cent. ore can be worked with a reasonable assurance of profit." The following values of ores

[4] In *Bulletin* No. 284, *United States Geological Survey*, p. 156 (1905).

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worked at various places have been reported:[5] Mt. Bischoff, Tasmania, 1.25 per cent. of Sn; Stannary Hills M. & T. Co., So. Australia, 3.12 per cent. of SnO_2; Lancelot Lode, Queensland, from 3 to 4 per cent. of SnO_2; Bolivia, "Under favorable conditions, [ores] as low as 3 per cent. may be worked at a small profit." · Dolcoath, 2 per cent. of black tin; Carnbrea and Tincroft Mines, Ltd., 1.5 per cent. of black tin. Once underground, the work of mining can be carried on the entire year without much difficulty. The season for milling the ore is necessarily very short—possibly five months at the most.

Table IV. gives the annual output of tin in various countries of the world during 1905 and 1906.[6] It is to be noted that the United States is conspicuous by its absence.

TABLE IV.—*The Principal Tin-Supplies of the World, During* 1905 *and* 1906. (*In Tons of* 2,240 *lb.*)

	1905.	1906.
English Productions	4,468	4,920
Straits to Europe and America	56,847	57,138
Straits to India and China	1,700	1,300
Australia to Europe and America	5,028	6,888
Banka sales in Holland	9,960	9,300
Billiton sales in Java and Holland	2,715	1,950
Bolivian arrivals on Continent Bolivian arrivals in England	12,500	14,700
Totals in long tons	93,218	96,196
Totals in metric tons	94,709	97,735

It is to be hoped that the tin-deposits in this section may prove to be of commercial importance. It must be said, however, that much time and money will be required to put the industry in Alaska on a paying basis. At the present time, operations are in the prospecting stage, and it is merely conjecture to say what will be the outcome.

ACKNOWLEDGMENTS.

In presenting this paper, I wish to express my gratitude to Prof. Jas. F. Kemp, and Mr. E. G. Spilsbury for valuable suggestions and criticisms. I should also mention specially the paper of Mr. F. L. Hess, entitled The York Tin Region,[7] in

[5] *Mineral Industry*, vol. xiv., pp. 538 to 546 (1905).

[6] *Mineral Industry*, vol. xv., p. 718 (1906).

[7] *Bulletin* No. 284, *United States Geological Survey*, pp. 145 to 157 (1905).

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which are described some things that I have independently observed.

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The Occurrence of Nickel in Virginia.

BY THOMAS LEONARD WATSON, BLACKSBURG, VA.

(Toronto Meeting, July, 1907.)

INTRODUCTION.

SULPHIDE ore-bodies of more or less lenticular shape occurring in metamorphic crystalline schists, gneisses, and slates, and conforming closely in strike and usually in dip to the inclosing rock, have been recognized and described along the Atlantic seaboard from Quebec to Alabama. The ore-bodies do not entirely conform, in all cases, to the structure of the inclosing schists, but they sometimes cut across the foliation in dip, though conforming more closely with the direction of strike.

In composition the principal sulphide mineral is usually pyrite or pyrrhotite, or both, accompanied by copper, frequently gold, and almost always some sphalerite and galenite. Other metallic ores are, so far as these sulphide bodies have been investigated, less common; silver is not uncommon, and in some of the pyrrhotite masses nickel is present in quantity sufficient to be commercially valuable. Cobalt frequently accompanies the nickel, usually in smaller amount, and the two vary much in relative proportion.

Quartz and calcite are probably the most common of the non-metallic minerals. A number of silicate minerals are frequent representatives in one place or another; garnet, amphibole, biotite, pyroxene, zoisite and others.

The sulphide bodies of Virginia, comprising both pyrite and pyrrhotite, are among the most extensive along the Atlantic seaboard. Among the pyrite bodies may be mentioned the well-known and extensively worked mines of Louisa and Prince William counties, located on the main pyrite belt (A, Fig. 1) of the middle and northern, eastern Piedmont region in Virginia, and among the pyrrhotite bodies is the famous "Great Gossan Lead" (B, Fig. 1) of the Floyd-Carroll-Grayson coun-

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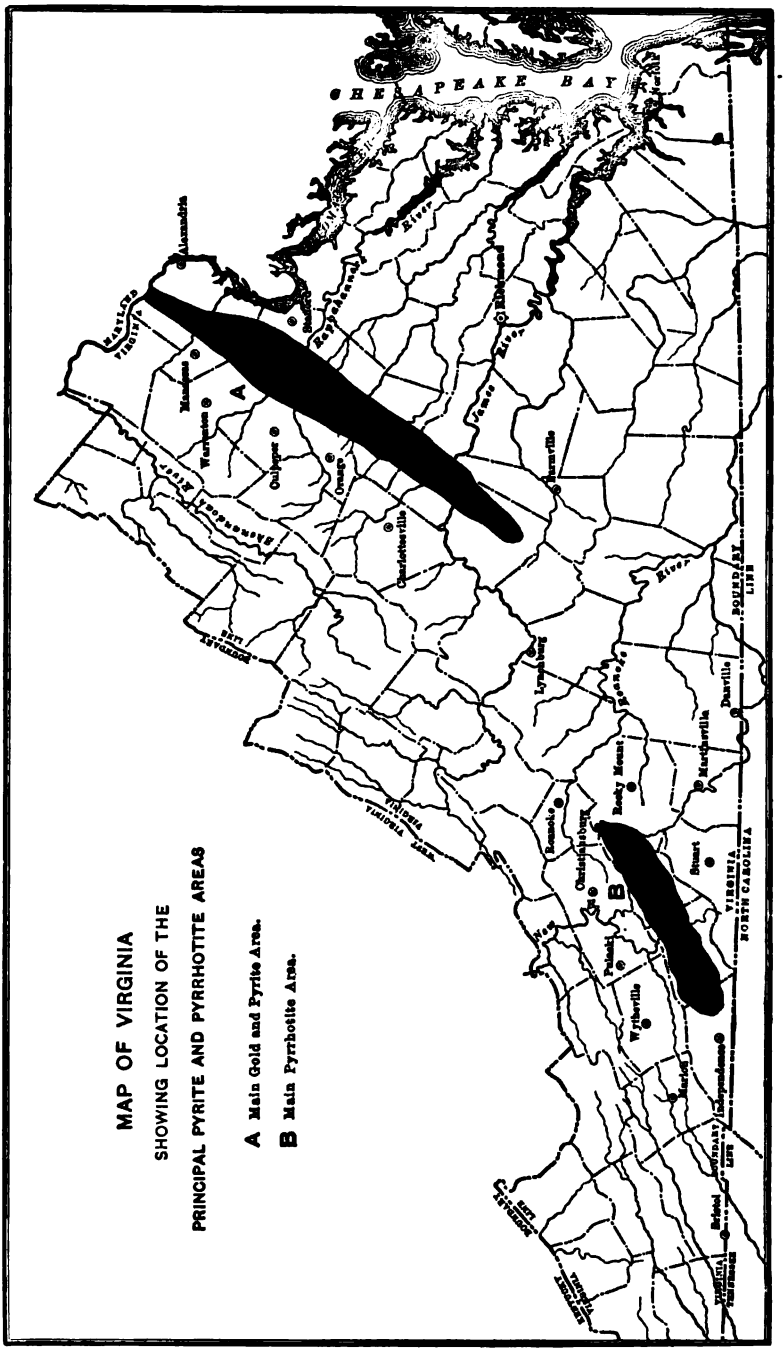

FIG. 1.—SKETCH-MAP OF VIRGINIA, SHOWING PYRITE AND PYRRHOTITE BELTS.

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ties plateau in southwest Virginia. Both nickel and cobalt, in minute quantities, are reported in association with the pyrrhotite bodies of the plateau region of metamorphic crystalline rocks in southwest Virginia, but, so far as I am aware, the working of them for either of these metals has never been seriously contemplated.

It is the purpose of the present paper to treat in some detail of a recently exploited body of niccoliferous pyrrhotite in the extreme northwestern corner of the Floyd county portion of the Floyd-Carroll-Grayson plateau, shown in Fig. 2. This deposit does not belong to either of the classes outlined above, but is associated with basic igneous rocks, the principal types of which are not unlike those of Sudbury, in Ontario, Canada.

Localities of Niccoliferous-Ore Occurrences in Virginia.

The existence of nickel in Virginia has been reported from a number of localities in the Piedmont region or crystalline area, especially in association with many of the extensive pyrrhotite bodies of the Floyd-Carroll-Grayson plateau in southwest Virginia, and in Amherst county, near and to the east of Lynchburg. In Amherst county, the pyrrhotite is somewhat sparingly developed as small grains and moderate-sized granular masses, in crystalline schists, partly hornblendic, of doubtful origin. More recently, nickel in association with peridotite masses has been reported from near Broadrun station, in Fauquier county, northern Virginia; but this locality has not yet been investigated.

Reliable analyses of the southwest Virginia pyrrhotite bodies, giving the exact percentages of nickel and cobalt, are unfortunately not available.

Developments and Value of the Ore.

The property described in this paper is controlled by the Virginia Nickel Corporation, which, from 1904 until recently, exploited the area under the name of Fidelity Exploration Co. Fig. 2 indicates openings made at four different places. Some of these are in schists and contain pyrite instead of pyrrhotite as the ore. This paper will be confined to the occurrence and genesis of the nickel at the Lick Fork openings.

This occurrence of pyrrhotite has been known for many years.

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The ore was dug and copperas made from it in a very crude way before the Civil War. The Lick Fork openings, Fig. 3, comprising shafts and drifts, were begun in the hard-rock out-

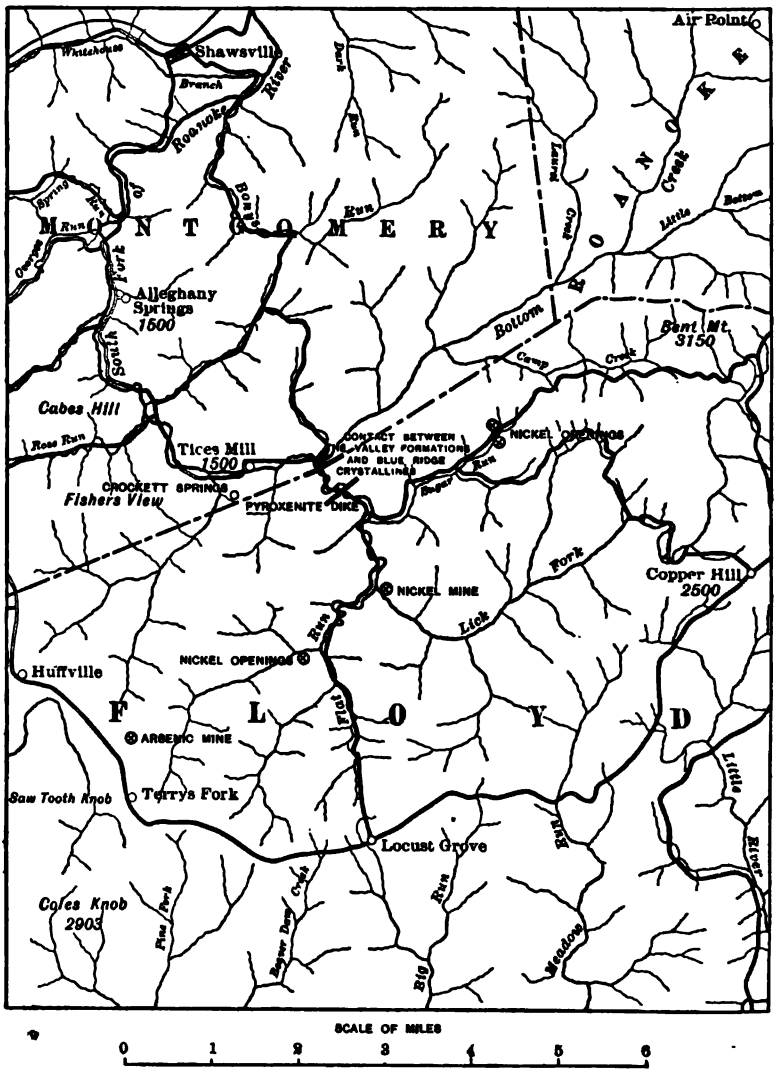

FIG. 2.—MAP, SHOWING LOCATION OF NICKEL- AND ARSENIC-MINES IN FLOYD COUNTY, VA.

crops at the base of a high and steep ridge, a few feet above the stream-level, and sunk less than 100 ft. below water-level. At the time of my visit, in May, 1907, a large amount of ore

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on the dumps was reported to average by actual assays not less than 1.75 per cent. of nickel, and a fraction of 1 per cent. of copper. As much as 0.4 per cent. of cobalt was reported; but the average is considerably less. Assays of the pyrite from other openings in the area are reported to yield from 3 to 4 per cent. of nickel.

Arsenopyrite (mispickel) is mined less than 4 miles southwest of the Lick Fork openings, but numerous analyses made of the ore do not show the presence of either nickel or cobalt.

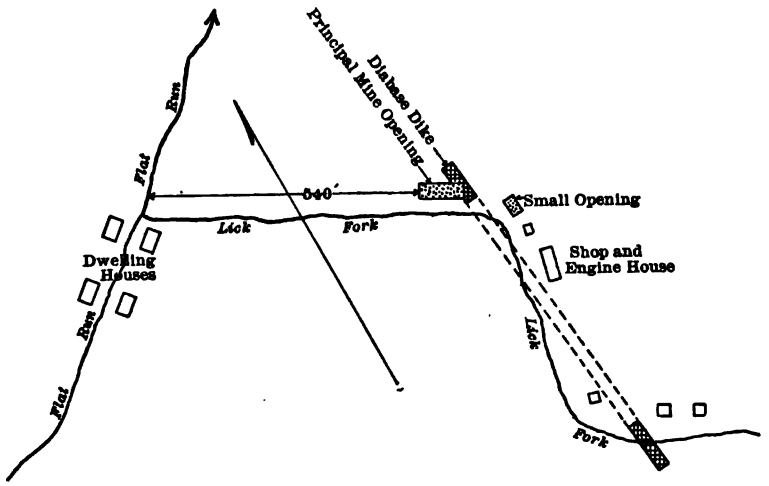

FIG. 3.—GROUND-PLAN, SHOWING LOCATION OF OPENINGS AT NICKEL-MINE, NEAR HEMLOCK, FLOYD COUNTY, VA.

LOCATION AND TOPOGRAPHY OF THE AREA.

Location.—As indicated on the accompanying map, Fig. 2, the area here described lies in the extreme northwestern part of Floyd county along the boundaries of Montgomery and Roanoke counties. The openings thus far made are wholly in Floyd county, the principal ones being directly on Lick Fork, within a few hundred feet of its confluence with Flat Run, the SE. headwaters of the South Fork of the Roanoke river. The Lick Fork openings are further located in a direct line from Shawsville, the nearest railway station, 7 miles nearly SSE. The openings are aligned along a nearly NE.–SW. direction for a distance of about 4.5 miles.

Topography.—Fig. 2 shows the area to be a part of the strongly dissected portion of the extreme NW. part of the Floyd-Carroll-Grayson counties plateau region. The main

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divide of the Blue Ridge is several miles further eastward, and the western margin of the Shenandoah limestone of the Valley region is less than 3 miles NW. in a direct line. Only one other mining-operation is located in the immediate area—namely, the mines and milling-plant of the U. S. Arsenic Mines Co., less than 4 miles SW. in a direct line from the principal nickel openings on Lick Fork. The mines of the arsenic company are practically on the divide between the north- and south-flowing waters of the plateau region, the nickel-bearing area being within the headwater-drainage of the north-flowing streams.

The area is one of strong surface-relief and of the mountainous type of topography, the higher ridges and knobs rising to elevations of from 2,500 to 3,000 ft. above mean tide-level. In the immediate vicinity of the principal openings on Lick Fork, the surrounding ridges rise about 600 ft. above the stream-level, the altitude of these openings, made at approximately local stream-level, being about 1,900 ft. The average elevation of the area is probably about 2,500 ft.

Geology and Petrography.

The area is composed exclusively of crystalline metamorphic rocks derived in part from original sediments and in part from igneous masses. It has been one of profound deformation, the results both of anamorphic and katamorphic changes being strikingly manifested in the rocks.

The rocks will be separately discussed under (*a*), the country-rock, and (*b*), the rocks immediately associated with the ore. The latter are igneous in origin, are much less altered from metamorphism, and are intruded into, and therefore later in age than, the country-rocks.

(*a*) *The Country-Rock.*—The country-rock is chiefly made up of a complex of micaceous quartz-schists and gneisses of variable composition, and usually of a pronounced thinly foliated type. Feldspar may or may not be a principal constituent of the schists; more often it is nearly or essentially absent. The gneisses, on the other hand, carry feldspar as one of the most abundant minerals. The country-rocks are undoubtedly derived, in part at least, from original sediments, while a part can be probably referred to altered igneous masses.

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Talcose and chloritic schists are found in places. East of the openings on Lick Fork, and within less than a mile along the stream, a medium-textured, highly quartzitic rock, composed largely of pale and deep blue opalescent quartz with some feldspar and a dark green pyroxene or amphibole, is exposed, and still further east fine-grained granite occurs. Toward the southwest end of the area and within a mile of the arsenic-mines the rock is a talcose schist in places. At the arsenic-mines still other rock-types are distinguished, the principal ones being a silvery white, very thinly foliated quartz-sericite schist, with occasional bluish quartz "eyes," and a feldspar-quartz (bluish opalescent quartz)—biotite gneiss, the origin of which is in doubt.

At the contact of the Palcozoic sediments of the Valley region with the crystallines, less than 3 miles NW. of the Lick Fork openings, the principal rock on the crystalline side is an irregular syenitic gneiss of thin foliation and highly feldspathic. It is porphyritic in places, the pinkish feldspar phenocrysts being usually elongated in the direction of the gneissic structure. Exposures of the variable syenitic gneiss are found at many points over the western part of the area, probably indicating a considerable belt of this rock.

(*b*) *Rocks Immediately Associated with the Ore.*—These are without exception of igneous origin, and range in composition from a pyroxene syenite (akerite, according to Brögger) to a very basic gabbro. Two distinct types of the gabbro are sharply defined in the Lick Fork openings—namely, olivine-diabase, and the ore-bearing gabbro proper. These are separately discussed below.

The structural relations of the syenite, diabase, and gabbro, as indicated in the larger opening on Lick Fork, are shown in Fig. 4. The diabase and gabbro seem to be later in age than the syenite, which they apparently penetrate in dike-like form. Evidence is lacking as to the exact age-relations of the gabbro or ore-bearing rock and the diabase. The diabase is in direct contact, as a hanging-wall, with the gabbro; has an average thickness, so far as the openings extend in depth, of from 8 to 10 ft., and an outer contact with the syenite, as its own hanging-wall.

In May, 1907, the openings were nearly filled with water, which prevented access to them and doubtless obscured much

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that might otherwise be cleared up from an underground study. I was reliably assured, however, that the contact between the gabbro and diabase was entirely sharp and without gradation of the two rocks into each other. It is entirely plain that since the last intrusion of igneous material the three rocks, syenite, gabbro, and diabase, have been subjected alike to intense dynamic strain, resulting in extensive fracturing, and otherwise altering them to some degree.

Pyroxene-Syenite.—Pyroxene-syenite, the rock into which the ore-bearing rock is immediately intruded, is, in its typical development, a coarse-grained dark grayish-green aggregate of feldspars and dark pyroxenes with, in places, a sprinkling of quartz

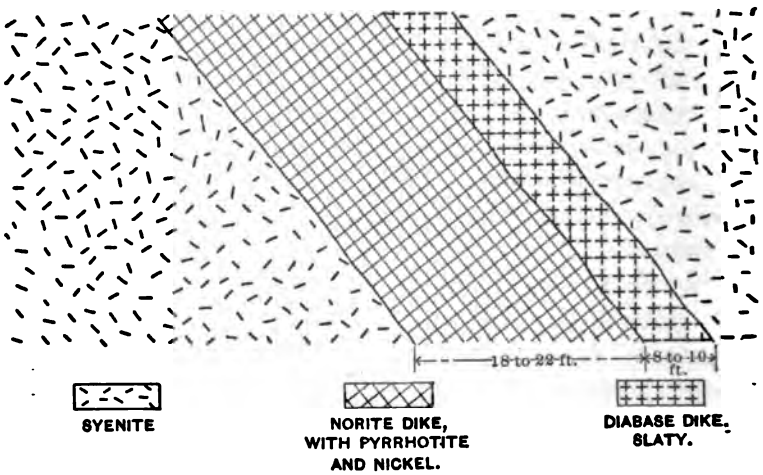

FIG. 4.—SECTION OF NICKEL-MINE, NEAR HEMLOCK, FLOYD COUNTY, VA. SHOWS STRUCTURAL RELATIONS OF THE SYENITE, DIABASE, AND GABBRO.

and biotite, and usually garnets. It is strikingly similar, both in hand-specimens and in thin sections, to the unakite-bearing rock of Madison county, Virginia, in the northern Blue Ridge region, described by Phalen as hypersthene akerite, and later traced by me through several adjoining counties in the same region.

The syenite has not been identified on the higher ridge-tops of the area, but is found exposed at considerably lower elevations on Flat Run and Lick Fork. It has not been definitely identified at a greater distance from the Lick Fork openings than two miles. The outcrops at different places vary in color and texture from the normal coarse-grained, dark, grayish-green rock, with only an occasional recognizable quartz-grain,

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to one of lighter color and from similar coarse to finer texture, in which a fair sprinkling of quartz occurs and a considerable development of garnet. In the finer-textured rock, biotite is frequently present in large amount.

In thin section the following minerals were noted: Orthoclase, plagioclase, microcline, pyroxene, biotite, hornblende, quartz, garnet, and the usual minor accessory minerals and alteration-products common to syenitic rocks. Pyroxene is the only essential ferromagnesian silicate, though biotite appears in nearly every thin section except two, sparingly in some but nearly equaling pyroxene in others, and hornblende is by no means rare. Feldspar is much the most abundant mineral, and is occasionally intergrown with quartz in micrographic structure. Microcline is present in most of the sections, and in many it is hardly less abundant than orthoclase and plagioclase. Plagioclase equals and in some sections exceeds the potash feldspar in amount, and the rock might perhaps with equal propriety be called a monzonite. In the absence of a complete chemical analysis, however, the name syenite is retained for the rock.

Analysis of a similar syenite (hypersthene akerite) from Milams Gap in the Blue Ridge of northern Virginia, analyzed and described by Phalen,[1] will give a general idea of the composition of the Floyd county syenite.

Hypersthene Akerite (syenite) from Milams Gap, Virginia.
W. C. Phalen, Analyst.

	Per Cent.
SiO_2,	60.52
Al_2O_3 (includes TiO_2),	16.99
Fe_2O_3,	0.60
FeO,	6.53
MgO,	1.59
CaO,	4.58
Na_2O,	2.83
K_2O,	3.91
H_2O,	0.88
P_2O_5,	0.74
MnO,	0.25
ZrO_2,	trace
Cr_2O_3,	trace
TiO_2,	n. d.
Total,	99.42

[1] *Smithsonian Miscellaneous Collections*, vol. xlv., p. 311 (1903).

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Extinctions measured against the twining lamellæ of the plagioclase indicate a feldspar of about acid labradorite composition. Evidences of strain are manifested both in the feldspar and in the quartz by frequent wavy extinction and fracturing, and in the former mineral by occasional bent lamellæ. The usual alterations of the feldspars and pyroxenes are noted, but are not important at this time.

Garnet is present to some extent in nearly every thin section, and in many it is considerable in quantity. Most of the hand-specimens of the rock show little or much of the mineral. It is entirely secondary, as its position in the sections indicates its derivation from the interaction of the feldspar and ferromagnesian minerals, usually pyroxene. It is present usually in large irregular granular masses and in every case it is greatly fractured. It is probably the common variety of calcium-iron garnet, andradite. The other minerals do not call for description here.

Fractures common to most of the minerals, a partly peripheral granulation of the light-colored minerals, occasional bent lamellæ of the plagioclase, and wavy extinction of the quartz and to some extent the feldspar, are microscopic evidences of strain in thin sections of the rock. Every outcrop of the rock shows extensive fracturing, the numerous irregular fractures crossing each other in nearly every direction, frequently breaking the rock into small irregular masses.

Olivine-Diabase.—So far as the opening extends in depth, the olivine-diabase occurs on the hanging-wall of the ore-bearing gabbro, in dike-like form, averaging in width from 8 to 10 ft. and separating the gabbro from the syenite. As indicated in Fig. 3, the dike is exposed in and crossed by Lick Fork after its first bend from the mine-openings, distant from the latter about 500 ft. These two neighboring exposures of presumably the same dike indicate a trend of N.20°E., and, as shown in Fig. 4, a dip of 45° to 60° ESE. Its micro-structure and composition are those of typical fine-grained olivine-diabase. Olivine is abundant in large micro-porphyritic grains and crystals. Apart from the plagioclase laths and the augite, a sparse sprinkling of iron sulphide occurs, apparently of primary origin.

Like the syenite, the diabase is extensively broken by fracture-lines, which cause it to break into small irregular, more or

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less rhombic, blocks when struck with the hammer. In the unweathered rock the diabase appears entirely massive, but specimens quarried from near or just above the water-level and exposed to the air for some time show a parting closely similar to slaty cleavage. In these weathered specimens of the rock, pyrite is visible to the naked eye, and some of it is of secondary origin, formed along the parting-planes mentioned above.

Mica-Gabbro (*Ore-Bearing Rock*).—Fig. 4 shows the relations of the ore-bearing gabbro to the syenite and olivine-diabase. It has an average width in the present main opening on Lick Fork of from 18 to 22 ft., and like the diabase it dips from 45° to 60° ESE.

It is a uniformly dark, dense, medium- to fine-grained rock, in which the somewhat abundant biotite shreds are easily recognized by the unaided eye. Megascopically it appears to be entirely made up of the dark silicate minerals and the metallic sulphides. The nature, distribution, and occurrence of the sulphides are described below. In the hand-specimens the rock appears entirely massive and without evidence of alteration of any kind.

Examination of a reasonably large number of thin sections shows the rock to be composed of a greater preponderance of the dark silicate minerals than of the lighter-colored ones. Orthorhombic and monoclinic pyroxene, biotite, hornblende, olivine, and plagioclase, are the principal rock-minerals. Not all of these are found in any single section. Pyroxene, biotite, hornblende, and plagioclase are in largest amount. Hornblende is variable in amount, being large in some sections and small in others, and olivine was noted in only one or two sections, but in these it was in quite appreciable amount. Monoclinic pyroxene is the most abundant dark silicate, and in some sections orthorhombic pyroxenes were not identified. Biotite is present in every section, usually in considerable quantity, as a primary constituent. Accessory orthoclase is present. Quartz and garnet do not occur.

Usually the minerals show strong cleavage-development and are more or less considerably altered to the usual secondary products, with the separation of much black oxide of iron from the ferromagnesian minerals. Minute fractures are common

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to all the minerals, and in many instances the biotite folia are bent and fractured across. Several sections showed chiefly areas of a fine mosaic of the minerals, especially the dark silicates, which undoubtedly represent mashing and granulation from pressure.

The Ores.

The ore is chiefly pyrrhotite, with finely disseminated chalcopyrite inextricably intermingled. It is fine-grained and massive, and apart from manifesting a tendency to break along planes of weakness it shows no evidence of banding. The sulphides both replace and inclose the rock-minerals at times.

Separate nickel-minerals were not observed. Indeed, no investigation was made to determine whether the nickel was present as an essential constitutent of the pyrrhotite, replacing in part the iron, or as a separate compound, as in the Sudbury district.

Mode of Occurrence of the Ores.

Megascopic.—The sulphides, pyrrhotite and chalcopyrite, chiefly the former, are quite freely distributed, but without striking regularity or uniformity, through the gabbro. In some parts of the rock the proportion of sulphides is very small; in others, they make up 50 per cent. and more of the total rock-mass, with all gradations between. The average ratio of sulphides to total rock-mass is considerably less than 15 per cent. It is not possible for me to say whether the ore-distribution favors any one zone, such as the foot- or hanging-wall, or the central portion of the rock, more than another. The exposures accessible to me in the openings were insufficient as data for a judgment on this question. A careful examination of the dumps, however, revealed some ore in nearly every piece of the ore-bearing rock, and I am reliably informed that the ore does not favor one zone in the rock more than another, but is distributed through the entire thickness of the ore-bearing rock—gabbro. The sulphides are distributed through the rock in small irregular areas, both connected and disconnected, and in stringers and veinlets, which penetrate the rock in all directions. These irregular small areas of ore may be composed entirely of sulphides, or the sulphides may inclose minute areas of the rock-minerals. Another occurrence is that of smaller isolated granular masses, resembling disseminations, such as

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FIG. 5.—Photomicrograph of ore-rock, showing sulphides (dark) between folia of biotite (medium gray), filling fractures and entering cleavages. Light-colored areas, altered and unaltered pyroxene. × 40.

FIG. 6.—Photomicrograph of ore-rock, showing sulphides (dark) rimming, penetrating and replacing pyroxene (white). × 40.

FIG. 7.—Photomicrograph of ore-rock, with ore (dark) replacing the iron-bearing silicate-minerals, chiefly pyroxene, with some biotite (white) along cracks and cleavages. Small unreplaced residues of silicate-minerals (white) in large area of sulphides (dark). × 40.

FIG. 8.—Photomicrograph of ore-rock, showing inclosures of biotite folia and pyroxene (white) in sulphides (medium gray). The dark area on right of figure is brownish-red chlorite derived from biotite. × 40.

FIGS. 5, 6, 7, 8.—MICROSTRUCTURES OF NICKEL ORE-ROCKS, FLOYD COUNTY, VIRGINIA.

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may ordinarily be observed in any basic igneous rock, and are usually regarded as original constitutents of the rock.

Microscopic.—Microscopic sections indicate, as remarked above, that the rock has undergone some alteration, although every section shows that most of the original minerals are remarkably fresh, except frequently around the margins and along the fractures which penetrate into and sometimes cross the grains, and in some cases along the cleavage-directions. Many instances are noted in which the original mineral is almost completely altered, but the ratio of such to the entirely or nearly fresh mineral is exceedingly small.

Microscopic sections show the relations of the sulphides to the rock-minerals to be quite similar, in some respects, to the Sudbury and the Rossland examples described by Dickson.[2] The sulphides probably show closer association with the dark silicate minerals pyroxene, biotite, and hornblende. The sulphides, filling fractures, are formed between the grains and along the cleavages, replacing the rock-minerals involved, either in whole or in part. These relations of the sulphides to the silicate minerals, indicated in Figs. 5, 6, 7 and 8, show the marked tendency of the sulphides to follow lines of weakness, such as along fracture- and cleavage-planes and between grains. In each occurrence more or less of the silicate mineral involved is usually replaced. Sulphide veinlets are frequently formed in the silicate minerals, especially the biotite, along the direction of cleavage (Fig. 5). Biotite is usually only slightly or not at all replaced by the sulphides, rarely entirely so, indicating, for those sections studied, at least, that it is one of the most-resistant minerals.

Frequently the sulphides show sharp outlines against the fresh silicate minerals, but where the latter are cleaved and broken the sulphides tend to occur along these directions. The relations of the sulphides to the rock-minerals, briefly stated above, are shown in Figs. 5, 6, 7 and 8. Attention is called to Fig. 7, which shows the massive ore inclosing small areas of unreplaced silicate minerals.

It is noteworthy that a very small proportion of the sulphides show relations to the silicate minerals such as to suggest that

[2] Dickson, C W., The Ore-Deposits of Sudbury, Ontario, *Trans.*, xxxiv., 25-64 (1904).

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they are primary and were among the first to crystallize. The proportion, however, is not greater than is ordinarily contained in a rock so basic as this one.

Genesis of the Ores.

The description given above, both of the ore and of the associated rocks, strongly suggests that the ore is chiefly of secondary origin. It occurs in an igneous rock, which in turn is intimately associated with others, and each shows visible evidence of deformation. The formation of most of the ore appears to have been later than the metamorphism of the rocks. In large part, the evidence for this is based on microscopic study, the results of which are briefly set forth above and need not be repeated here.

Concerning the source of the ore it is yet impossible to make a definite statement. Several possibilities present themselves, which, for lack of sufficient data, are not discussed, but, so far as there is evidence at hand, it is probable that the ore was the work of ascending solutions derived from great depth.

Conclusions.

In summary the essential facts developed in this paper are;

I. That the general area is one of crystalline metamorphic schists and gneisses, derived in part from original sediments and in part from igneous masses.

II. That the ore-bearing series of rocks, which is of igneous origin and intrusive into the schists and gneisses, comprises syenite, diabase, and gabbro. Gabbro is the ore-bearing rock and is in contact with syenite on the foot-wall and with diabase on the hanging-wall. Each of these types shows evidence of some metamorphism.

III. That the ore, averaging not less than 1.75 per cent. of nickel and comprising pyrrhotite chiefly, and a little intermingled chalcopyrite, is largely secondary and was formed after the metamorphism of the rocks. Microscopic sections show, from the relations of the sulphides to the rock-minerals, that the sulphides followed principally the lines of weakness in the silicate minerals, with considerable replacement of them.

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The Present Source and Uses of Vanadium.

BY J. KENT SMITH, PITTSBURG, PA.

(Toronto Meeting, July, 1907.)

VANADIUM is generally spoken of as a rare element; but, even in the light of our resources as known a couple of years ago, this description could be accepted in a qualified sense only. In fact, vanadium is very widely distributed, being a constituent of most clays, while even caustic soda has been shown to contain almost regularly a minute proportion of it, although the quantity in each case is so small as to render impracticable the extraction of the metal therefrom with profit.

The opening-up of a large deposit of vanadium sulphide ore in South America has put a new phase on the commercial outlook of vanadium, the technical value of which for the improvement of steel has been proved by rigorous scientific investigation.

This sulphide ore has the following analysis:

	Per Cent.
Vanadium sulphide,	39.84
Molybdenum sulphide,	1.57
Nickel sulphide,	1.49
Iron sulphide,	4.07
Arsenic sulphide,	nil.
Copper sulphide,	nil.
Manganese sulphide,	nil.
Free sulphur,	30.57
Phosphorus,	nil.
Silica,	13.60
Alumina,	2.46
Combined water, Carbonic acid,	5.00
Lime, magnesia, potash, soda and unaccounted for,	1.40
	100.00

Being a free-burning ore it is calcined with ease, losing 45 per cent. of its weight in the process. The calcined ore has the following analysis:

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	Per Cent.
Vanadic oxide (V_2O_5),	58.08
Iron oxide,	4.98
Molybdenum oxide,	2.62
Nickel oxide,	2.24
Arsenic oxide,	nil.
Copper oxide,	nil.
Manganese oxide,	nil.
Phosphoric acid,	nil.
Silica,	25.00
Alumina,	4.52
Lime, magnesia, potash, soda, etc.,	2.56
Sulphur, 0.23 per cent.	
	100.00

The deposit is situated in Peru, South America; mining-conditions are easy, and the ore is transported to the port of Callao by direct railway-communication extending almost the entire distance. The vanadium claims have an area of 5.25 sq. miles, being 3.5 miles long by 1.5 miles wide. A large proportion of this area is vanadiferous, some probably owing to the impregnation of the aluminous earth by solutions of oxidized vanadium. Five distinct veins of large dimensions have already been proved on the property. The vein at present being worked has a width of 16 ft., and has been opened up on the face for a distance of 200 ft.; this vein dips at an angle of 65°, and by means of driving a tunnel lower in the face of the hill its existence in its pristine richness has been proved 140 ft. back. The working is done by means of "levels."

Previously the most commercial sources of vanadium at our disposal were the Spanish lead-ores. These, as mined, contain only 4 or 5 per cent., but by primitive processes of hand-dressing could be concentrated so that the "heads" contain, say, 12 per cent., of vanadic oxide. An analysis of this class of ore (vanadinite) is as follows:

	Per Cent.
Vanadic acid,	11.49
Lead oxide,	34.15
Lead sulphide,	1.43
Ferrous oxide,	13.17
Manganese oxide,	0.77
Zinc oxide,	0.62
Molybdenum oxide,	trace
Calcium carbonate,	0.74
Siliceous matter,	35.67
Silver, 3.5 oz. per ton.	

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The ore usually contains a little more lead and less siliceous matter, and, in addition to the components shown, my invariable experience has been that such ores contain copper, arsenic, and phosphorus. Sometimes the percentage of copper rises considerably; *e.g.*, in chileite we have:

	Per Cent.
Vanadic oxide,	13.50
Lead oxide,	54.90
Lead chloride,	0.30
Copper oxide,	14.60
Arsenious oxide,	4.60
Phosphoric acid,	0.60
Sand and clay,	2.00
Iron peroxide, alumina, etc.,	3.50
Lime,	0.50
Loss in ignition,	2.70

A much rarer mineral, descloizite, containing a considerable amount of maganese in place of the copper, has been found in a crystallized condition and in comparatively small quantities. Its analysis shows:

	Per Cent.
Vanadic oxide,	22.46
Lead oxide,	54.70
Zinc oxide,	2.04
Copper oxide,	.90
Manganous oxide,	5.32
Manganese dioxide,	6.00
Sand and clay,	3.44
Iron peroxide, alumina, etc.,	1.50
Loss in ignition,	2.20

On comparing these analyses, the value of the sulphide-deposit will be seen at once; since, in addition to the practically inexhaustible quantity of the mineral obtainable, its richness and the comparative ease with which the vanadium can be separated (as opposed to the separation of smaller quantities from complex ores, the treatment of which involves the removal of lead, copper, arsenic, phosphorus, and zinc, in addition to the siliceous gangue), make it a great commercial consideration.

Vanadium is a silvery white metal of high melting-point—said to be 2,000° C., and, at all events, higher than the melting-point of platinum—so that its commercial use in the metallic form is practically restricted to its use as a refractory metal (in the filaments of electric lamps, for instance). An alloy of iron with vanadium in the proportion of two parts of iron to one part of

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vanadium has a melting-point of 1,375° C., or one more than 100° C. below that of mild steel; and it is in this form that the metal is marketed for the use of the steel-maker.

The old custom of judging a steel by its resistance to static load, and the amount it would stretch under such load, gave a certain general guide as to the behavior of metal under conditions of engineering practice which are now rapidly being left behind. Even under these conditions, mysterious failures occasionally occurred, which, it is now evident, had their origin in the inability of the metal to resist strains applied in a totally different manner from that in which it was tested.

As the requirements of modern engineering construction became more and more drastic respecting the power of resistance to rapidly-repeated strains and shocks, accompanied, of course, by increased demands as to actual strength, the old test-conditions were still further receded from in practice; and it became necessary to resort to alloy-steels.

Here some success was scored; but the true requirements were nevertheless lost sight of, and the metal was still judged almost entirely by its behavior under static loads. With the same ductility, an increase in the strength was looked upon as the one desirable thing, although it is now recognized that this improvement in strength, if attained at the expense of dynamic properties in the original steel, is, in most cases, of comparatively little use to the engineer. In short, we lost ourselves in straining after something which we did not want, and which we attained only at the expense of something we did want.

Since, in modern machine-construction, and especially in those parts which are liable to failure in use, we require, above all, superior resistance to repeated stresses, to alternating stresses, to simple, repeated or alternating impacts, and to fatigue (which is the outward and visible sign of intermolecular vibratory deterioration), a new field has been opened, and in this field vanadium has been found, by extended experiment and practical experience, to be uniquely valuable. As to static properties, vanadium intensifies greatly the strengthening effect of another ingredient, such as chromium, thus enabling the proportion of the latter to be so reduced as to avoid "poisoning" the metal with regard to desired dynamic qualities. Moreover, vanadium is itself highly contributory to the dynamic excellence of mild

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TABLE I.—*Properties of Vanadium-Steel Compared with Some Other Steels.*

Test.	For Comparison. 1. Carbon 'Axle' Stock.	For Comparison. 2. Nickel 'Axle' Stock.	Dynamic. 3. Vanadium Axle Steel.	Combined. 4. Vanadium Crank-shaft Steel.	Static. 5. Vanadium Continual Mesh Gear Steel.	Nature.
Yield-point, lb. per sq. in.....	41,330	49,270	63,570	110,100	224,000	Static.
Ultimate stress, lb. per sq. in..	65,840	87,360	96,080	127,800	232,750	Static.
Ratio.....	0.62	0.56	0.66	0.87	0.96	Static.
Contraction of area, per cent..	61	58	61	58	39	Static.
Elongation, per cent. on 2 in...	42	34	33	20	11	Static.
Tortional twists.....	2.6	3.2	4.2	2.5	1.8	Static.
Alternating bends	10	12	18	10	6	Intermediate.
Pendulum impact, ft.-lb.....	12	14	16.5	12	6	Dynamic.
Alternating impact. Number of stresses.	960	800	2,700	1,850	800	Dynamic.
Falling weight on notched bar Number of blows.	25	35	69	76	.....	Dynamic.
Rotary vibrations Number of revolutions.	6,200	10,000	67,500	.....	.....	Dynamic.

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steel. By retarding segregation, it renders the metal particularly susceptible to enormous improvement by tempering and heat-treatment generally. By virtue of this characteristic, steel can be "automatically" rendered highly resistant to wear and abrasion. Again, vanadium toughens steel; confers upon it great power of resistance to torsional rupture; in a word, endows it with increased "life" in practical use.

Being a powerful "medicine," vanadium is used in small doses only. In engineering-steels, the maximum proportion required seldom exceeds 0.2 per cent. By its judicious use, combinations are thus made possible which could not be formed without it, and which permit the successful fulfillment of complicated requirements, whether chiefly static, chiefly dynamic, or divided more or less equally between the two classes.

These varying and partially contradictory demands must all be considered, in order to meet modern conditions; and vanadium is the only substance yet known to us, by means of which they can be successfully satisfied. How it meets them is shown in Table I., which contrasts the properties of vanadium-steel with some other steels. The data presented in this table were obtained under like conditions of testing.

The great industrial importance of vanadium being thus evident, the question of its commercial supply becomes a vital one. As already observed, it was, until recently, classed among the "rare" metals, not because it was not known to be widely distributed, but because it could be profitably produced in few localities and small quantities only. The development of the large deposit of rich vanadium sulphide, above noted, and the erection in Pittsburg, Pa., by the American Vanadium Co., which owns that mine, of works having a daily capacity of 2,000 lb. of vanadium-alloys, settles the question of commercial supply.

In the preparation of these brief notes, I am indebted to Mr. J. O. Handy, of Pittsburg, for assistance in analytical work.

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The Presence of Gold and Silver in Deep-Sea Dredgings.

BY LUTHER WAGONER, SAN FRANCISCO, CAL.

(New York Meeting, April, 1907.)

HAVING given in a former paper[1] the results of assays of sea-water, bay-mud, dredgings from San Francisco bay, etc., and believing it might be interesting to extend the work to include some deep-sea dredgings, I procured from the Smithsonian Institution six samples taken by the U. S. steamer *Albatross*, and marked as follows:

No.	Station.	Locality.	Latitude.	Longitude.	Depth in Fathoms.
1.	2103.	Off Delaware bay.	38° 47′ 20″ N.,	72° 37′ W.,	1,091.
2.	2265 to 2297.	Between Chesapeake bay and Hatteras.	35° 37′ N.,	74° to 75° W.,	49 to 70.
3.	2420.	Off Chesapeake bay.	37° 03′ 20″ N.,	74° 31′ 40″ W.,	104.
4.	2528.	East of Georges bank.	41° 47′ N.,	65° 37′ 30″ W.,	677.
5.	2572.	SE. of Georges bank.	40° 29′ N.,	66° 04′ W.,	1,769.
6.	2681	South of Nantucket.	39° 43′ N.,	70° 29′ W.,	990.

The above samples were assayed by the cyanide-method, described in my former paper (p. 807), about 30 g. of sample being used, and the following results were obtained:

Value in Milligrams Per Metric Ton.

No.	Station.	Gold.	Silver.
1.	2103.	145	1,014
2.	2265 to 2297.	44	304
3.	2420.	15	353
4.	2528.	267	1,963
5.	2572.	125	377
6.	2681.	66	414

No. 4, Station 2528, consisting of red clay and volcanic ash, was re-assayed; 35 g. was roasted at a low heat for one hour and weighed 34.05 g. after roasting; the color was changed to a deep brick-red; and when assayed by the cyanide-method

[1] The Detection and Estimation of Small Quantities of Gold and Silver, *Trans.*, xxxi., 798 to 810 (1901).

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the material was found to contain 94 mg. of gold and 496 mg. of silver per metric ton, from which it appears that roasting is not beneficial. This sample, and Nos. 3 and 6, were also assayed by fusion with lead upon coal (using 500 mg. of sample), and gave much larger amounts of silver, from which it may be concluded that the cyanide does not extract all of the silver.

The general result of the assays indicates that the bed of the Atlantic, whether coastal or pelagic, carries appreciable amounts of gold and silver, and that the deep-sea bottom is relatively richer in gold than that nearer the shore line.

I desire to thank Mr. L. E. Aubury, State Mineralogist of California, and his staff, for the use of the laboratory of the California State Mining Bureau, and for aid rendered during the investigation.

Quantitative Field-Test for Magnesia in Cement-Rock and Limestone.

BY CHARLES CATLETT, STAUNTON, VA.

(Toronto Meeting, July, 1907.)

THE rapid development of the Portland-cement industry implies that the country is being very carefully searched for material suitable for its manufacture. Such material can be found at a great many places; but deposits thoroughly satisfactory in chemical composition, and at the same time meeting all the other requirements connected with the economic production and profitable marketing of the product, are exceedingly few, if I may judge from the investigations which I have had to make in a number of States.

The raw mixture which by its clinkering and grinding forms Portland cement contains approximately 75 per cent. of carbonate of lime. This is found in nature as limestone or marl. The natural rock commonly carries either more than the required percentage of carbonate of lime (in which case it has to be reduced by suitable admixture of clayey ingredients) or less—in which case purer lime carbonate has to be added. The controlling ingredient, therefore, is carbonate of lime, or limestone of the right composition.

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Among the most serious difficulties is that of finding limestones low enough in magnesia to meet the exacting requirements of certain specifications. All engineers recognize that there is a limit beyond which it is not safe to let the magnesia go; but within that limit there is considerable difference of opinion. Until that difference is finally settled, no one would be willing to invest the large sum of money necessary for a Portland-cement plant, if it were not certain that the usual requirements in this particular could be satisfied.

The usual limit in well-established specifications is 5 per cent. of magnesium oxide in the finished cement, but many specifications limit the amount to 3 per cent. These figures correspond roughly to 3.2 per cent. and 1.9 per cent., respectively, of magnesium oxide in the raw mixture.

It is comparatively easy with a little practice to detect magnesia to the extent of 10 or 15 per cent. The appearance of the stone, the way it effervesces with dilute hydrochloric acid, its hardness, etc., settle this question and enable us to eliminate a great many otherwise promising deposits. It is not possible to form by superficial examination any estimate of the amount of magnesia when it is as low as 5 or 6 per cent.; yet it would be hard to overestimate the advantage of being able to determine in a few minutes, and at a point remote from a laboratory, whether a material will probably produce a cement well within the limits in this respect, and whether one is justified in securing options and taking detailed samples for further investigation. The value of such a determination would not be destroyed by the fact that it was not exact, since the purpose of a preliminary investigation would be fully met if a difference of 1 per cent. in magnesia could thus be clearly and easily detected. It would then be possible to eliminate those rocks which contained decidedly too much magnesia, and to retain, for further and more exact laboratory investigation, those which were very low in magnesia, or might possibly be low enough to meet the requirements of the case.

As the result of considerable investigation, undertaken with a knowledge of the conditions involved, my assistant, Mr. J. J. Porter (a member of the Institute, and recently elected associate professor of metallurgy at the University of Cincinnati), worked out the following method, which I have personally

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found valuable in this connection. The method is based on the fact that while calcium hydrate is very soluble in a solution of cane-sugar, magnesium hydrate is only slightly so. This gives us a means of precipitating magnesia in the presence of lime.

Reagents.—1:1 hydrochloric acid: 1 volume hydrochloric acid of 1.20 specific gravity plus 1 volume of water. A 30 per cent. solution of potassium hydrate: 30 g. of potassium hydrate (pure by alcohol) to 100 cc. of water. This must be free from carbonate. Sugar-solution: A cold saturated solution of granulated sugar. Calcium carbonate: C.P. precipitated. Standard limestones as needed, powdered to 40-mesh size.

Apparatus.—A small steel mortar for powdering samples; small sieve, 40-mesh size; measuring-spoon, holding between 0.4 and 0.5 g. of 40-mesh limestone powder, level full; 12 test-tubes with mark at 10 cc.; stand to hold 12 test-tubes in two rows of 6 each; a pipette, 1.75 cc., for HCl; a pipette, 1.5 cc., for sugar-solution; a pipette, 1 cc., for potassium hydrate; 6 small funnels; filter-papers to fit, uniform size; test-tube holder; alcohol-lamp; water-bottle; test-tube brush.

Method.—Crush sample, powder in mortar, and sift. Pour the powder into measuring-spoon and level off by knife-blade or card. Transfer the measured portion to test-tube.

Add from pipette 1.75 cc. of HCl, and, when effervescence has nearly ceased, boil for a moment. Then add enough pure calcium carbonate to neutralize the excess of HCl. Boil until steam issues freely from the mouth of the test-tube, in order to drive out all carbon dioxide. Add water to the 10-cc. mark, and mix thoroughly by shaking the test-tube.

In another test-tube, place, by means of the proper pipettes, 1.5 cc. of sugar-solution and 1 cc. of potassium-hydrate-solution, and dilute with water to the 10-cc. mark. Mix by shaking. Then filter the solution of the stone, allowing the filtrate to run into this alkaline sugar-solution. If magnesia is present, a precipitate of magnesium hydrate will form at the line of contact of the two solutions. After the filter has drained, these solutions should be mixed by inverting the test-tube.

The density of this precipitate is roughly indicative of the percentage of magnesia in the stone, and by comparison with standard stones run in parallel this percentage can be estimated pretty accurately, the probable error not being greater

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than 20 per cent. of the amount of magnesia present. As in all comparison methods, some practice is necessary to insure satisfactory determinations. Since the magnesium hydrate is not entirely insoluble, and some magnesia is apt to remain in the insoluble portion of the sample, the results are low. This probable error is desirable, since it is better to send samples to the laboratory for further examination than to discard something which might be satisfactory.

Notes and Precautions.—The fineness of the material affects the weight held by the measuring-spoon and also the ease of solution—hence the need of sifting.

There should be sufficient HCl to dissolve thoroughly all the carbonates of the rock; and this condition is assured if the resulting solution is yellow from ferric chloride. At the same time, it is desirable to have the smallest possible excess of HCl, because the presence of the alkaline chlorides decreases very materially the delicacy of the test.

The object of adding calcium carbonate is two-fold. It precipitates the iron; and it creates a condition of uniformity as to the neutrality of the solution which is very essential for comparative results.

It is desirable that all the filter-papers used in a series of tests be of uniform size and quality of paper, so that they may retain a uniform amount of the rock-solution.

The method of adding the reagents is important, and should be closely followed.

After trying various methods, the one given above has been found to give the most satisfactory results, both as to delicacy and reliability. It is, of course, essential that the potassium hydrate contain no carbonate, and it should therefore be occasionally tested with barium hydrate or with a solution of lime in sugar.

The magnesium hydrate precipitate frequently comes down with a greenish color. The cause of this is uncertain; but it is most prominent in those samples containing organic matter. Possibly it is due to a trace of ferrous iron produced. This color causes the precipitate to appear somewhat more dense and opaque, and, therefore, should be allowed for in making comparisons. With a little practice this is not difficult, particularly if the comparison is made by transmitted light.

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The scheme outlined above does very well for rocks carrying up to 4 per cent. of magnesium oxide. Above this, the precipitate gets too dense for comparison without further dilution.

It is desirable that the pipettes should be made from glass tubes of different sizes, and that short pieces of rubber tubing should be attached as mouth-pieces.

All of the apparatus can be conveniently packed in a box of the inside dimensions of 10.5 by 7 by 6.75 in.

A double top, in the form of a shallow box, of a size to fit snugly in the larger one, may be used as a convenient receptacle for the test-tubes, serving by means of auger-holes as a test-tube stand.

Small splinters from the larger pieces making up the permanent sample may be taken for this examination, and can be crushed without danger of loss by direct pressure and a grinding motion rather than by blows.

Such small pieces, if treated with dilute acid in a test-tube, will usually disintegrate entirely if the carbonate of lime is considerably in excess of the theoretical requirement, thus indicating the need of the admixture of clay or shale for the manufacture of cement.

If the action of the acid ceases before all of the carbonate is dissolved, and recommences when the splinter is rubbed so as to remove the clayey covering, the stone is apt to need the addition of purer limestone to produce a satisfactory mixture.

If the splinter retains its shape when treated with dilute acid, but all of the lime carbonate is dissolved, the stone probably approaches closely a natural Portland-cement mixture.

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Geology of the Virginia Barite-Deposits.

BY THOMAS LEONARD WATSON, BLACKSBURG, VA.

(Toronto Meeting, July, 1907.)

I. Historical.

Barite has been mined for many years in various parts of Virginia, probably the earliest mining-operations being in Prince William county, within 600 ft. of the Fauquier county line, about 4 miles south of east from Catlett, a station on the Southern Railway. It is claimed that the mineral was mined here as early as 1845.

Mining-operations in Campbell and Pittsylvania counties in the Piedmont region, and near Marion in Smyth county, in the southwest Virginia Valley region, were begun about 30 years ago. The mining and milling of barite on a commercial scale in Tazewell and Russell counties are more recent, and commenced about 15 years ago.

II. Geographical and Geological Distribution.

Barite occurs in many counties in the State, as shown in Fig. 1, but the industry has been confined to only a few of them. Its occurrence is noted in two of the three major divisions of the State—namely, the Piedmont region east of the Blue Ridge, and the Paleozoic area west of the Blue Ridge. In the Piedmont region the mineral has been mined in the following counties: Bedford, Campbell, Louisa, Prince William, and Pittsylvania. In the region west of the Blue Ridge, composed of Paleozoic sediments, barite has been mined in Montgomery, Russell, Smyth, and Tazewell counties. Of the counties mentioned, Bedford, Campbell, and Pittsylvania of the Piedmont region, and Russell, Smyth, and Tazewell of the Valley region, have been the principal producers. In 1906, operations were confined to five counties—Bedford, Louisa, Pittsylvania, Russell, and Tazewell—but in the first two counties named there was no production, the work being solely development.

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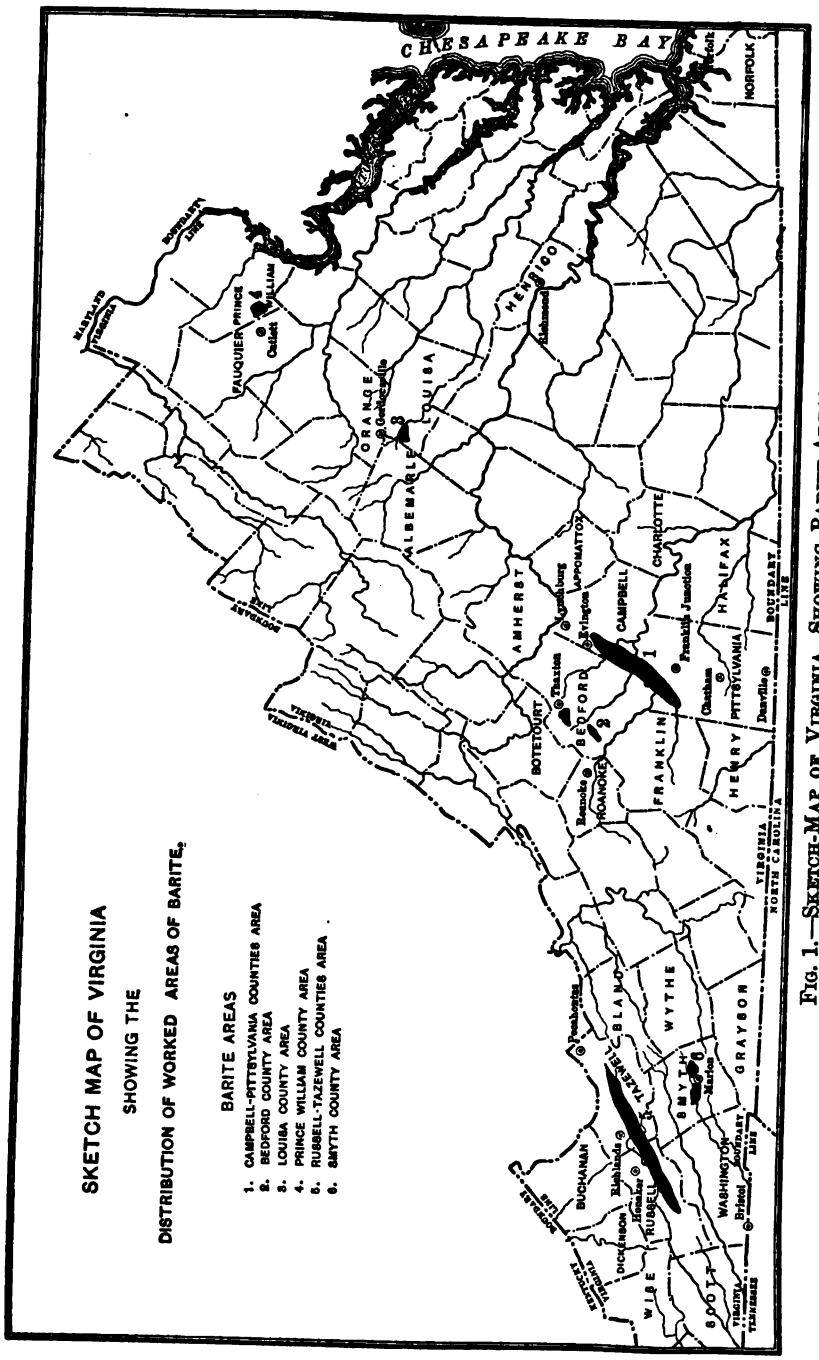

FIG. 1.—SKETCH-MAP OF VIRGINIA, SHOWING BARITE-AREAS.

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Geologically, the barite-deposits of Virginia may be grouped into three unlike areas: (1) those deposits of the red shale-sandstone series of Triassic age; (2) those of the crystalline metamorphic area, the age-relations of the rocks being unknown, but probably pre-Cambrian for the most part; and (3) those of the Valley region, associated for the most part with the Cambro-Ordovician limestone (Shenandoah or Valley) or its residual decay. Areas (1) and (2) compose the Piedmont province, which stretches eastward from the Blue Ridge to the fall-line or the western margin of the Coastal Plain sediments.

III. General Mode of Occurrence.

The Virginia barite-deposits of commercial importance thus far developed, which have been worked for barite alone, are associated with limestone as pockets or lenticular masses, largely in the nature of replacements, and as vein-like masses filling fractures in the limestone. In southwest Virginia and elsewhere in the Valley region where the barite occurs in the limestone, it is often found as superficially loose lumps and nodules of irregular shapes and sizes imbedded in the residual clays derived from the limestone. In several instances the barite is not associated with limestone, but occurs directly in siliceous crystalline rocks removed, so far as we know, some distance from limestone. The barite-deposit near Thaxton, in Bedford county, best illustrates the occurrence of barite in siliceous crystalline rocks remote from limestone-masses.

In Campbell and Pittsylvania counties of the crystalline area the barite is intimately associated with coarsely crystalline limestone (marble) and its residual decay. The local differences in the mode of occurrence of barite in Virginia are best brought out in the description given below of the three geologically-unlike areas in which deposits are found.

IV. The Triassic Area.

As yet only one deposit of barite of commercial importance has been developed in the numerous areas of Triassic rocks occurring in Virginia east of the Blue Ridge. About 4 miles south of east from Catlett station, in Prince William county, and within 600 ft. of the Fauquier county line, barite has been mined at different times since 1845. It was last worked in

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1903, with a production of 1,500 tons of ore. The opening of this deposit probably marks the first mining of barite in Virginia.

The ore was mined by shafts and open cuts, the greatest depth reached in mining being 108 ft. The grinding and preparation of the ore for market were conducted in a mill built for that purpose on the property. This mill was afterwards burned, and in its place now stands a partly-completed crushing-house, located near the main shaft.

The area forms a part of the eastern Piedmont region, characterized topographically by a gently undulating surface, without marked relief. The geological position of this deposit is within the eastern margin of the red shale-sandstone series of the Triassic area which crosses the Potomac river west of Washington and terminates about 10 miles south of Culpeper, the Virginia portion of the New York-Virginia area as defined by Russell.[1]

The rocks of the immediate barite locality consist of ferruginous red sandy shales and a light-colored crystalline limestone. Measured at numerous points west of the mine the dip varies from 10° to 15° west of northwest. Between Catlett station and the mine the shales are penetrated by occasional masses of diabase.

Much of the material composing the dumps at the mine is a limestone breccia, in which red shale fragments are cemented by an impure crystalline limestone. Fractures are frequent, and are filled with barite and occasionally with calcite-crystals. These facts suggest that the barite-deposit occupies a crushed or fractured zone in the Triassic sediments, induced probably by faulting, although no evidence for such is apparent on the surface. In view of these facts, and since faulting is a characteristic structure of the Triassic areas in Virginia and of the similar eastern areas in general, it seems reasonable to ascribe the brecciation in the Prince William county locality to this cause.

The barite is associated with both the red shales and the impure limestone, usually as a deposition-product from the solution which filled fractures in the red shale. The widest of the

[1] Russell, I. C., Correlation Papers—The Newark System, *Bulletin* No. 85, *United States Geological Survey*, p. 20 *et seq.* (1892).

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barite-filled fractures, reported to be from 4 to 8 ft., form the chief source of the minable mineral. The barite occurs also as thin tabular cleavable masses in the limestone. It is of good white grade, both finely and coarsely crystalline, massive, and, judging from the ore on the dumps, quite free from most of the common impurities, especially manganese. I did not have access to the shafts, but it is reliably reported that the association of ore with limestone increases with depth.

V. The Crystalline (Piedmont Plateau) Area.

The Virginia-Piedmont province forms a part of the eastern crystalline region which extends southwestward from New York to northern central Alabama. Its limits in Virginia are from the Blue Ridge on the west to the fall-line, western margin of the Coastal Plain, on the east; and it widens southward. Excepting the Triassic areas the rocks are all crystalline, and comprise greatly altered sedimentary and igneous masses. The region is made up of a complex of schists, gneisses, and granites, with, in places, interfoliations of slates, quartzites, and limestones. This complex is further intersected by intrusions of basic eruptive rocks belonging, so far as they have been studied, to the diabasic, dioritic, and gabbroic types. The bulk of the rocks composing this region, the oldest in the State, were mapped by the older geologists as Archean, but more recent studies reveal the fact that a part of them are as late as Ordovician.

The occurrence of barite has been noted in nine counties in the crystalline area, but the principal production has been from Campbell and Pittsylvania counties, with Bedford next in point of production. The ore has been mined in these counties for 30 or more years.

1. *The Campbell-Pittsylvania Counties Area.*

Beginning in the middle western portion of Campbell county, several miles east of Evington and about 15 miles south of Lynchburg, a belt of barite-deposits is traced southwestward to 3 or more miles south of Sandy Level in the northwestern part of Pittsylvania county, a distance of about 50 miles. Numerous openings have been made at different points on the belt, many of which have been extensively worked and have pro-

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Fig. 2.—Barite-Deposits of the Bedford-Campbell-Pittsylvania Counties Area.

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duced large quantities of excellent ore. Fig. 2 is a sketch-map of the belt, showing the location of the various openings made.

This area, occupying a part of the middle western Piedmont region, presents no unusual features in topography from that of the Piedmont in general.

The most extensively-worked deposits on the belt are grouped about two centers, Evington in Campbell county, at the NE., and Toshes and Sandy Level in Pittsylvania county, at the SW. extremity of the belt. Operations were begun in the two counties within a short time of each other, with probably the Hewitt mine in the vicinity of Evington, which dates back to 1874, the first to open. The mines in the vicinity of Toshes and Sandy Level in Pittsylvania county were opened at least 25 years ago, and in both counties the mines have been operated almost continuously from the beginning.

(*a*) *Mode of Occurrence.*—Two characteristic occurrences of barite are observed in the Campbell-Pittsylvania area, always in association with each other, and equally as strongly emphasized in one part of the area as in the other. The first and principal occurrence is in intimate association with the crystalline limestone as irregular lenticular bodies or pockets, which measure from 100 to 200 ft. or more, replacing the limestone. At the Hewitt mine in Campbell county some of the barite-pockets were reported entirely inclosed by the limestone. The barite observes the same coarsely-crystalline massive structure as characterizes the limestone, and in several places gradation of the barite into the limestone was observed.

For the depths so far attained, there is immediately below and above the limestone, a variable thickness of a nearly black clayey mass, usually preserving the foliation of the original rock from which it was derived, and colored black from manganese oxide, derived from the decay of a limestone-schist. Through this black clayey mass are usually distributed, in irregular fashion, lumps and nodules of barite of large and small size. Occasionally, barite stringers of slight thickness are formed along the foliation-planes of the clay. Figs. 3 and 4 illustrate the two occurrences of barite here described.

(*b*) *Associated Minerals.*—Calcite in the form of the coarsely-crystalline marble with which the barite occurs is much the most abundant associate. Pyrite and chalcopyrite are fre-

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quently present as thin stringers, and as disseminated small grains and crystals in some of the barite; but more especially in the limestone. These are usually intermingled, and in several instances a green staining of malachite has been observed

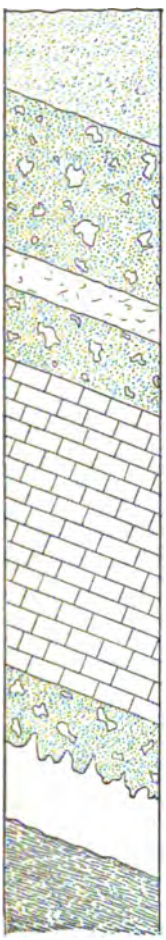

FIG. 3.—COLUMNAR SECTION, BENNETT BARYTES-MINE, PITTSYLVANIA COUNTY.

from the alteration of the chalcopyrite. Manganese oxide and iron oxide are frequent associates in places, but, as a rule, they are not noticeable in the best grades of the barite. These are usually more abundant and, therefore, more troublesome in the barite mined from near the surface. A small amount

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of the barite from the Hewitt mine in Campbell county is reported to have been highly charged with manganese oxide. Tremolite occurs both in the black clay and in the fresh crystalline limestone, and in places much biotite accompanies the tremolite in the latter. Biotite also occurs quite freely distributed through portions of the limestone in places not in association with tremolite.

(*c*) *Associated Rocks.*—The principal rocks of the Campbell-Pittsylvania barite-area are crystalline schists with intercalated thin beds of coarsely-crystalline limestone. The schists are of

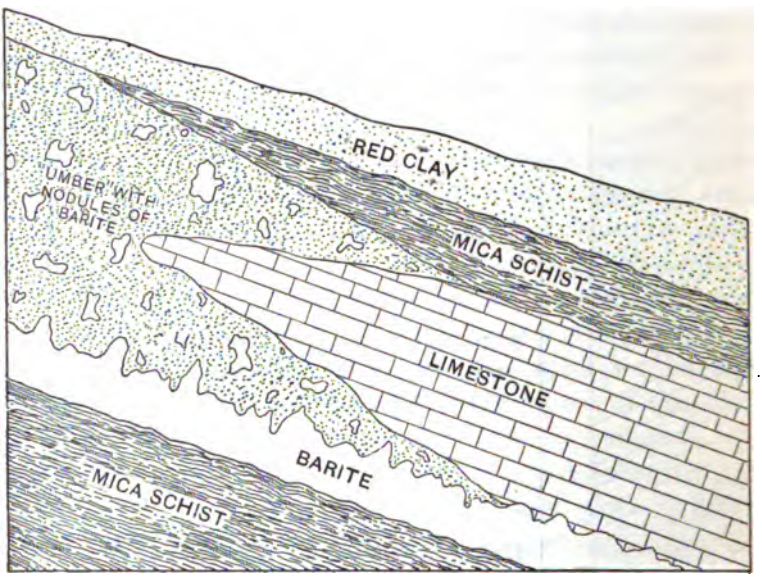

FIG. 4.—IDEAL SECTION IN BENNETT BARYTES-MINE, PITTSYLVANIA COUNTY.

two distinct types—mica-schist and quartz-schist. The immediate rocks with which the barite is associated are marked by the essential absence of feldspar. The schists are composed of mica with minimum quartz, and they are unquestionably derived from sediments. Igneous rocks of basic composition occur in the vicinity of Toshes, and an irregular gneiss of granitic composition is found within 0.75 mile NE. of the Bennett mine, the origin of which, whether sedimentary or igneous, has not been determined.

In the Evington portion of the area, the underlying rock immediately in contact with the limestone-masses on the north-

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west side is a fine-grained quartzite-schist of considerable purity, and containing small bright scales of white mica developed mostly along the planes of schistosity. The overlying rock immediately in contact with the limestone-masses on the northeast side is a variable mica-schist, always of fine texture and thinly foliated. Variation is from a moderately fine-grained muscovite-biotite schist at the Saunders-Phillips mines to a very fine-grained, lustrous, sericite-schist at the Hewitt mine. A fine-textured mica-schist, heavily charged with minute grains and crystals of black magnetite, forms an additional facies of the schist at the Hewitt mine. The mica-schists are composed essentially of mica without feldspar.

The mines developed on the southwest end of the belt in the vicinity of Toshes indicate an inclosure of the limestone-masses by a much coarser textured biotite-muscovite schist on the two sides. Feldspar is only recognized as a scantily-developed constituent of the rock, but thin stringers of quartz are interleaved at times with the schist, and small dike-like bodies of a coarse crystallization of pink feldspar and quartz frequently cut across the foliation of the mica-schist. The kaolinized equivalents of these dike-like masses are found in the same position in the residual clays derived from the schist.

Where exposed in mining, the limestone is a coarsely crystalline massive marble of considerable purity in places. In color it is generally white, occasionally pink, and sometimes greenish. It contains more or less of the silicate minerals, biotite and tremolite, and is frequently charged with pyrite and chalcopyrite. Manganese oxide and iron oxide are noted in places. The limestone is not of uniform thickness, but thins and thickens, the maximum observed thickness being about 60 ft. It conforms in dip and strike to the structure of the inclosing schists described above.

In composition the limestone is composed essentially of calcium carbonate with small amounts of magnesium carbonate, as shown in the analyses below, made by Dr. Walter B. Ellett, of specimens which I collected:

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	1. Per Cent.	2. Per Cent.	3. Per Cent.
Insoluble matter,	1.66	0.87	1.10
Alumina, Iron oxide,	0.24	0.30	0.96
Barium sulphate,	0.62	0.65	1.62
Calcium carbonate,	89.36	93.33	91.07
Magnesium carbonate,	6.61	2.82	3.73
Copper sulphide,	trace	trace	0.36

1 and 2. White crystalline limestone from the Hewitt mine, Campbell county. 3. White and pink crystalline limestone from the Ramsay mine, Pittsylvania county.

A third and important type of rock intimately associated with the barite, and whose fresh equivalent is a limestone-schist, is a black manganiferous and ferruginous clay, locally called "umber." It is found at every opening made on the belt, and its position is next to the limestone, occurring, as a rule, on both sides. Openings made near the outcrop usually penetrate a considerable thickness of the black clay, but do not, as a rule, encounter the limestone. When followed down, however, for a short distance in the direction of the dip, the limestone appears inclosed on either side by the dark clay, which apparently thins on depth and ultimately disappears. Like the associated schists, the black clay always contains the foliation-planes preserved in it of the original limy schist from which it was derived.

Leached mica-folia and small partly-oxidized areas of light green tremolite are found in the black clay of the Pittsylvania county mines. The relations of the clay to the limestone-masses and the mica-schist, together with its structure and composition, reasonably support the belief that it was derived by decay from a limy schist which was transitional between the well-defined limestone on the one hand and the mica-schist on the other. The kaolinized equivalents of the pegmatite dike-like forms found cutting the schists are observed in similar position in the black clay, as shown in Fig. 3. Much good barite is mined from the black clay, in which it occurs imbedded as nodular masses. Figs. 3 and 4 make clear the above relations of the clay to the limestone and the schist.

Samples of the black clay which I collected from the Bennett mine in Pittsylvania county, and analyzed by Dr. Walter B. Ellett, gave the following results:

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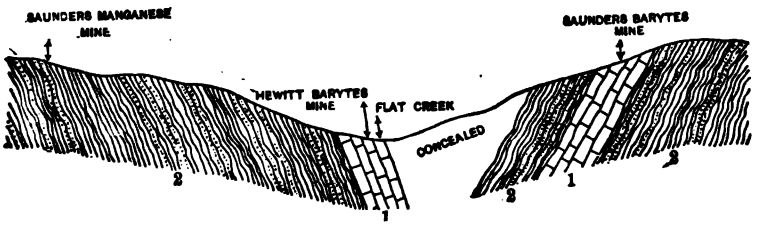
	Per Cent.
Insoluble residue,	14.20
Alumina,	4.96
Ferric oxide,	32.40
Manganous oxide,	19.49
Lime,	2.06
Magnesia,	trace
Barium oxide,	trace
Copper,	trace

(*d*) *Structure.*—The rocks are all schistose and preserve a general NE. strike with local variations, which, so far as measured, range from N. 30° to 55° E. The pure limestone masses are more or less massive, becoming decidedly schistose with de-



1. Limestone; 2. Mica-quartz schists.

FIG. 5.—SECTION ABOUT 1.5 MILES EAST OF EVINGTON, CAMPBELL COUNTY. (Section About 3 Miles Long.)

creased purity. Greater variations are shown in the dip of the rocks.

At the northern end of the belt, near Evington, in Campbell county, at the Saunders-Phillips mines, the schists dip N. 60° to 65° W. At the Hewitt mine, which is about 2 miles S. 70° W. from the Saunders mine, the dip is toward the SE. and quite steep, affording a distinct synclinal structure, as shown in Fig. 5. Near the southern end of the belt at Toshes, in Pittsylvania county, the openings at the Bennett mine show a variable dip to the SE., which is much flatter than in the Campbell county area, the probable average being about 20°. Outcrops of the schist at other places in this vicinity gave concordant results in dip. As indicated in the measurements of dip on the two ends of the belt, the folding has not been of uniform intensity. At the northeastern end the folding is steeper and of a more closed type, while at the southwestern end it is flatter and of a more open type.

(*e*) *Mines.*—In Campbell county the principal mines are the Hewitt, Saunders, Phillips, and Anthony, grouped near together

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and within a few miles east and SE. of Evington. Of these, the Hewitt mine has been the most extensively worked. It is located on the west side of Flat creek, about 2.5 miles from Evington, and about the same distance from the Saunders and Phillips mines. It was worked almost continuously from 1874 until 1904, when it was abandoned on account of water. It is developed by numerous shafts and drifts, the greatest depth reached in mining being about 160 ft. This depth extends below the local water-level, making it necessary to pump the water from the openings, which was troublesome, and finally led to suspension of work. The strike of the rocks is N. 55° E., with a steep SE. dip. The limestone with which the barite is associated has a thickness of about 60 ft., and is underlain by a quartzite-schist on the northwest side, locally designated the foot-wall, and overlain by a thinly foliated mica-schist on the northeast side, locally called the hanging-wall. As described above, the ore occurs, here and elsewhere in this belt, as irregular bodies replacing the limestone, and as irregular rounded nodules and masses in a black manganiferous and ferruginous clay, locally called umber.

The Saunders and Phillips mines adjoin each other, and are located on the crest of a well-defined NE.–SW. ridge on the east side of Flat creek, about 3 miles east of Evington. The developments comprise shafts, pits, and tunnels, the deepest one of which does not exceed 100 ft. The first openings were made about 20 years ago. The rock associations are identical with those described at the Hewitt mine. The strike of the schist is N. 30° E. and the dip N. 60°–65° W. About 2 miles SW. of the Saunders mine is the Anthony mine, which has produced a considerable quantity of ore.

The barite area in the extreme northwest corner of Pittsylvania county has been more extensively worked than any other part of the belt. It has been developed by a large number of mines, grouped in two nearly parallel belts on either side of Pig river, just south of its entrance into Roanoke river. Beginning at the northeast end the easternmost belt, trending approximately NE.–SW., has been developed by the following mines: Berger, Ramsay, Bennett, Parker, Thompson, and Dryden Wright. This belt lies approximately 0.75 mile east of Toshes. The westernmost belt is developed by the Tom Wright

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mine, 1 mile east of Sandy Level, and by the Hatchet, Meas, and Davis mines SW. of Sandy Level.

The Thompson mine is reported to have been the first one opened in the area; followed by the Parker, Berger, Bennett, and Ramsay, in the order named. The first four of these were worked more than 25 years ago, and the barite produced from each mine was very large in quantity and excellent in quality. Except the Bennett mine, which is operating at present, the others have been idle for some years, and very little could be seen at the time of my examination in September, 1906.

The mines in this area were developed by numerous shafts and drifts and some open work. The greatest depth yet reached in mining is 120 ft., the depth of the working-shaft at the Bennett mine. A description of this mine, one of the most extensively worked in the district, and the only one in operation in 1906, may be taken as typical of the area.

The rock-succession at the Bennett mine is shown in Fig. 3, which represents a vertical section of the 120-ft. shaft. As shown in this sketch, and also in Fig. 4, the wall-rock is coarse-grained, thinly-foliated mica-schist, intersected by pegmatite dikes composed of a coarse crystallization of feldspar and quartz. The fresh mica-schist is exposed in the bottom of the shaft, where it is penetrated for a depth of 20 ft., the overlying rock being a black manganiferous and ferruginous clay, usually thinly foliated, and derived from a lime-schist. A similar black clay also underlies the limestone, occurring between it and the fresh mica-schist. The limestone, which is a coarsely crystalline marble and charged to a small degree with both sulphide and silicate minerals, is 40 ft. thick, and occurs between the two layers of black clay.

The barite is associated with both the limestone and the black clay, in part as a replacement of the limestone, and as irregular, rounded masses and nodules in the over- and under-lying clays. The largest concentration of the barite is between the limestone and the schist, and it has an average thickness of about 10 ft. The contact between the barite layer and the limestone is very irregular, the ore often penetrating far into the limestone, as shown in Figs. 3 and 4. The contact between the barite and the mica-schist is sharply contrasted with that made with the limestone, and is sharply defined and quite

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regular. The barite is massive-granular, moderately coarsely crystalline, and of good white color. Five grades of the ore are made and marketed.

As indicated on map, Fig. 2, numerous other openings have been made between the Evington group of mines and the Toshes-Sandy Level group. Of these, perhaps the Maddox mine is one of the most important if not the most important. It is located about one mile SW. of Otter river station. Barite of excellent quality and in large quantity was mined, but the mine has not been operated in recent years on account of water, which makes the mining too expensive.

2. *Bedford County Area.*

Barite occurs and has been mined at a number of different points in Bedford county. Several mines near the Campbell county line have yielded large quantities of the mineral. In the western part of the county, between Bedford City and Roanoke, a recent operation shows an interesting occurrence of the mineral. The deposit lies about 8 miles NW. from Thaxton, and is reported to have been first opened in 1866, when a small quantity of the barite was shipped to Baltimore. It was re-opened again during the fall of 1906. The barite occurs in a completely-schistose coarse-grained granite filling a fracture. Some distance away from the fracture the granite is entirely massive and porphyritic, the feldspar phenocrysts being of large dimensions.

As nearly as could be determined, the fracture has a N. 10° to 20° E. course, and dips about 60° SE. An open cut about 20 ft. deep has been made along the course of the fracture for a distance of about 450 ft. The granite in the vicinity of the fracture is deeply decayed, although the fracture is distinctly shown at one end of the cut where it carries no barite.

The barite is crystalline, and varies in color from white to deep blue-gray. In places, much galenite in small grains and occasional sphalerite are disseminated through the barite. It seems quite probable that the source of the barium has been from the feldspar of the granite. The associated galena and sphalerite indicate that they were deposited contemporaneously with the barite. Whether they were introduced as soluble sulphides and deposited as such, or whether they were intro-

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duced as sulphates and subsequently reduced to the sulphide form, there is no evidence.

3. *Louisa County Area.*

Barite occurrence and mining in Louisa county are limited to the Walker place, 0.75 mile south of Mechanicsville, and 3 miles south of east from Lindsay, the nearest railway-point. The openings comprise a number of test-pits and several shafts, the deepest one of which is between 70 and 80 ft. Mining had been temporarily suspended for several months prior to my visit and the openings were filled with water, which prevented entering them for study of the ore- and rock-relations.

The area, which forms a part of the crystalline region east of the Blue Ridge, is topographically a nearly flat, gently undulating surface, averaging about 500 ft. above mean tide-level.

The rocks are metamorphic crystalline schists of probable sedimentary origin. They are very thinly foliated micaceous schists, considerably altered, the foliation-planes of which are so regular and closely spaced as to be called, locally, slates. They strike approximately NE.–SW. and observe a general southeast dip, with a probable average of about 45°. Exposures of the rock are rare because of the considerable depth of residual decay, chiefly gray and red clays. Quartz-fragments, both large and small, frequently litter the surface, indicating quartz-veins or vein-like masses interleaved with and cutting across the foliation of the schists. Careful search failed to indicate the presence of limestone, and close inquiry further confirmed its absence.

Examination of the ore was necessarily confined to the dumps. The ore is a moderately white grade of coarsely crystallized barite, remarkably free from impurities other than the usual discoloration from the red iron oxide. Several lumps of the ore showed cavities filled with nearly perfect quartz-crystals. Drused surfaces of large tabular barite-crystals are abundant. The ore was traced from the surface downward in the openings, observing a general but variable dip toward the southeast. It is reported to be pockety in mode of occurrence, widening and narrowing sharply and frequently, but having a thickness of about 3 ft. where worked in the deep shaft.

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Evidence is apparently lacking for regarding the ore as a replacement-deposit. It probably represents a filling of an irregular fracture in the crystalline schists, the barium salt of which was probably derived from some mineral or minerals, possibly feldspar, composing the surrounding rocks.

VI. The Valley (Paleozoic) Region of Southwest Virginia.

The Virginia Valley region lies west of the Blue Ridge and comprises a vast thickness of Paleozoic sediments, the principal member of which is the Valley or Shenandoah limestone, separable into several divisions, of Cambro-Ordovician age. Barite occurs in a number of counties in the middle and northern parts of the Valley, but mining of it in the Valley province has been confined largely to Russell, Smyth, and Tazewell counties in southwest Virginia. Wherever commercial deposits of barite have been opened in this province they have been found in association with the Shenandoah limestone or its residual decay. Within recent years, mining-operations have been limited to Tazewell and Russell counties.

1. *Mode of Occurrence.*

In southwest Virginia the barite is in association with the Shenandoah limestone or its residual decay. It observes certain minor variations of occurrence from place to place. It fills in part, at least, fractures in the limestone, and in part it replaces the limestone. These occurrences of barite in the limestone are fairly well shown in the southeastern part of Wythe county. Here the mineral associations with barite are limonite, sphalerite, galenite, pyrite, and occasional fluorite, in certain openings named below, from which iron- and zinc-ores have been mined. In the Tazewell-Russell counties area the common associates are limonite and calcite, with some siderite, and occasional fluorite.

In addition to its occurrence in the fresh limestone, the barite is found as small and large nodules irregularly distributed through the red clay resulting from the decay of the limestone. Variation in local occurrence and in mineral association is brought out in the following description of the individual areas.

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2. *Wythe County Area.*

In the southern part of Wythe county, and near the eastern margin of the Shenandoah limestone, barite occurs in association with the metallic ores in some of the zinc- and iron-mines. No attention has been given to the mining of barite in this locality, nor is it known whether commercial deposits of the mineral exist.

In several of the brown-iron-ore pits at Ivanhoe, barite has been observed in some of the limestone pinnacles as irregular porous or cellular masses intimately associated at times with pyrite, sphalerite, and galenite. These latter minerals, sulphides, were noted in several instances as inclosures in the barite, and the barite replaces in part the limestone. Northeast of Ivanhoe, at the Bertha zinc-mines, which have been operated for several years for iron-ore, barite is found in places as loose nodules imbedded in the red clay derived from the limestone decay.

About 0.5 mile NE. of Bertha, at the Barren Springs iron-ore pits, barite is again found similarly occurring. The mode of occurrence and the associations of the barite in the above localities suggest the introduction of the barium salt along fracture-lines in the limestone, and part replacement of the limestone by barite. Moreover, there is every reason for regarding the barite, the sulphides, and the little fluorite found as having been brought in and deposited at the same time. Whether the sulphides were carried as soluble sulphides, or as sulphates and deposited as sulphides, there is no convincing evidence.

3. *Smyth County Area.*

Barite was extensively mined some years ago near Marion, the county-seat of Smyth county. Mining and shipping of barite in this county were begun about 1877, and the work was continued more or less energetically until about 1885. Some mining on a small scale has since been done from time to time. Barite-mining in this county was largely confined to an area about 3 miles west from Marion, with but a small amount of material mined on the east side of the town. Most of the barite mined was prepared by several local mills, the greater part being prepared by the mill at Marion.

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The mining of barite was largely confined to the lumps and nodules of the mineral imbedded in the residual red clays derived from the Shenandoah limestone. The barite nodules were often mixed with cherty masses and broken or detached pieces of limestone. The mining did not reach 100 ft. in depth. In some instances mining extended into the fresh and hard limestone. Gradation from the barite into the limestone was shown, and in such cases much care and labor were necessary to separate the two. The exposures of limestone in some places showed more or less barite mixed with it.

4. *The Russell-Tazewell Counties Area.*

Extensive deposits of barite are found in Russell and Tazewell counties, these being the largest producers of barite at present in the State. Geographically, the distribution of the barite in these two counties is chiefly along the southern slope of Kent Ridge and its prolongation NE.–SW. along the valley

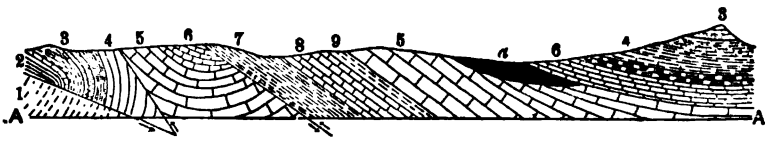

FIG. 6.—STRUCTURE SECTION ALONG LINE A—A OF FIG. 9, SHOWING STRUCTURAL RELATIONS OF THE BARITE AND ROCKS. *a* IS BARITE. Adapted from *Tazewell Folio, U. S. Geological Survey.*

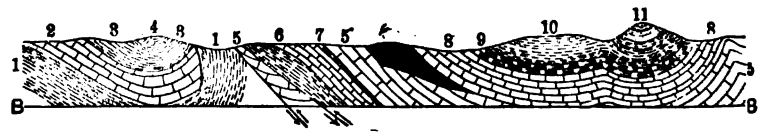

FIG. 7.—STRUCTURE SECTION ALONG LINE B—B OF FIG. 9, SHOWING STRUCTURAL RELATIONS OF THE BARITE AND ROCKS. BLACK IS BARITE. Adapted from *Tazewell Folio, U. S. Geological Survey.*

of Clinch river, extending from near North Tazewell to near Lebanon, a distance of more than 30 miles, partly shown in Fig. 9.

Barite has been mined at numerous points along this NE.–SW. belt, the principal mines being near North Tazewell; 3 miles south of Richlands; 3 miles from Honaker on the Clinch river; and on the southwestern end of the belt near Lebanon.

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Throughout this belt the barite is found in the upper portion of the Knox dolomite and its residual decay. Sections, Figs. 6 and 7, show the structural relations of the Knox dolomite and the adjacent rocks on the northwest and southeast, near Sword creek and Richlands, in Tazewell county.

The barite occurs as small and large lumps of irregular shapes assembled in the residual clay of the limestone, and in pocket-form and vein-like bodies filling spaces in the limestone, and in part replacing the limestone. Fig. 8 illustrates one of the principal modes of occurrence of barite in this area. The barite is crystalline, of good white quality, and in most places is quite free from impurities. The mines of the Clinch Valley Barytes Co., near Honaker and Gardner, show a coarsely-crystalline white barite, in platy, more or less radiate masses—

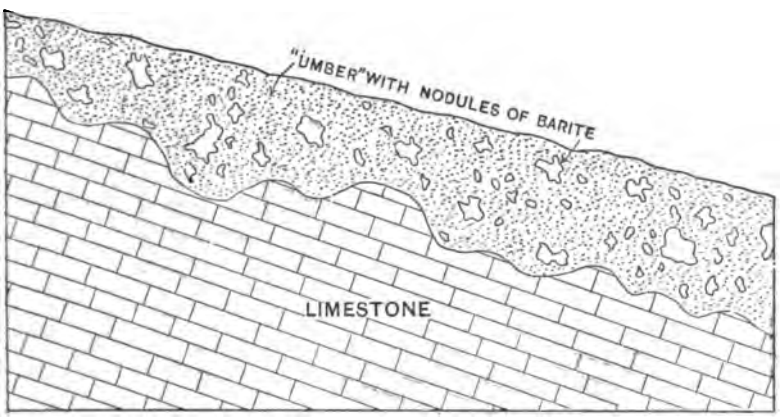

FIG. 8.—PRINCIPAL MODE OF OCCURRENCE OF BARITE IN RUSSELL AND TAZEWELL COUNTIES.

a structure which is strongly emphasized on weathered surfaces of the ore. The commonest impurity at these mines is iron oxide, mostly in the form of limonite. Manganese oxide occurs but sparsely, or not at all. In one of the pits a little violet fluorite and small fragments of green chert were noted in association with the ore.

The greatest depth attained in mining is 103 ft. at the mines of the Pittsburg Baryta & Milling Corporation on the northeast end of the belt. Most of the mining done has been for the lump or nodular ore occurring in the limestone clays, won from shallow open pits and cuts. Some hard-rock mining in the

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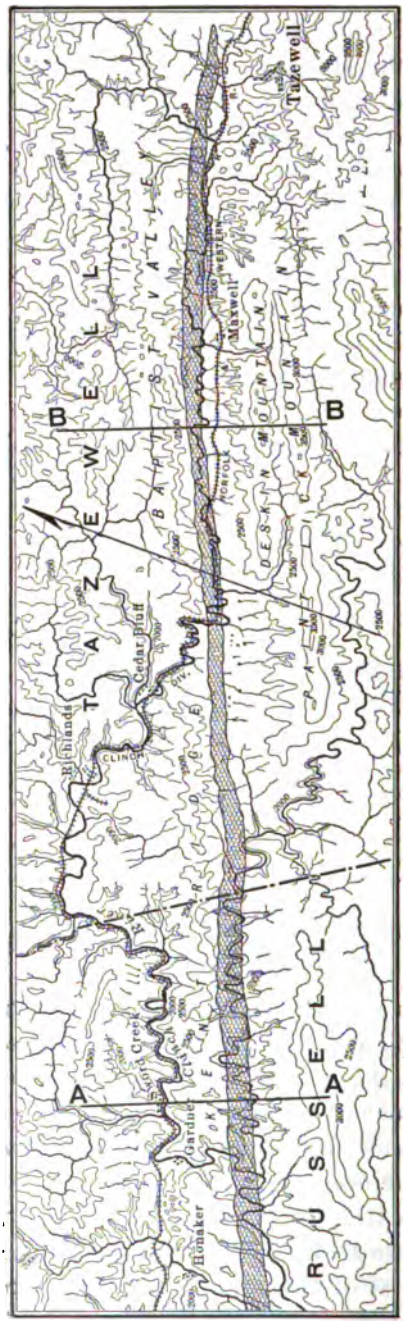

Scale, 0.25 in. = 1 mile, approximately.
Contour interval, 500 ft.

FIG. 9.—MAP OF A PART OF TAZEWELL AND RUSSELL COUNTIES, SHOWING POSITION OF GREATER PART OF BARITE ZONE. CHECKERED AREA THROUGH CENTER OF MAP IS BARITE. Adapted from the Economic Geology Sheet of the *Tazewell Folio, U. S. Geological Survey.*

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limestone has been done in places. It is probable that in the future much limestone-ore will be mined, since large bodies of excellent grade material are exposed at several places along the belt, especially on the properties of the Pittsburg Baryta & Milling Corporation.

At the NE. end of the belt an ore-body 4 ft. wide has been mined to a depth of 22 ft. and for a distance of more than 300 ft. The ore has been tested to a depth of more than 100 ft. without indications of its exhaustion. Preparations are being made to mine this ore-body to a greater depth. The barite occurs in vein-like bodies in the limestone, is very white, free from impurities, and makes the best "snow-flake" grade manufactured by the company. On the extreme southwest end of the belt, near Finney, the properties of the same company show ore-bodies in the limestone of the same white and otherwise excellent grade of ore as that described above on the northeast end. Three miles south of Richlands this company has mined more than 6,000 tons of barite from its property. Unlike the ore described above on the two ends of the belt, that south of Richlands is lump ore mined from the red limestone clays.

VII. Genesis of the Barite-Deposits.

The work of F. W. Clarke, Sandberger, and others, demonstrates the wide distribution of barium in rocks, usually present only in minute traces, but not infrequently in appreciable quantity. In the absence of necessary chemical work on the Virginia rocks in which the barite-deposits are found, but from the field-character and relations of the deposits, it seems reasonable to assume that the source of the barium was largely if not entirely from the rocks in which the deposits are now found. There is no evidence in support of a deep circulation.

The barium of the Valley barite-deposits is believed to have been derived from the Shenandoah limestone, the rock in which the deposits are found. Of that forming the deposits east of the Blue Ridge in Piedmont, Virginia, it is not possible to say whether the barium was derived from the crystalline schists or from the associated limestone-masses, or from both. It is reasonably certain, however, that the source of the barium in the deposit near Thaxton, Bedford county, was the silicate minerals of the granite, probably the feldspar.

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The barium was probably liberated and carried in solution as the soluble bicarbonate when, under proper conditions, it was precipitated as the insoluble sulphate. Some recent laboratory experiments made by Dickson with solutions of barium carbonate on selenite-crystals and pure anhydrite in the presence of CO_2, and on pyrite-crystals in the presence of an oxidizing agent, H_2O_2, resulted in each case in the precipitation of barium sulphate. The presence of pyrite in the Virginia rocks may suggest the possibility of its connection as a precipitating-agent in the formation of the barite-deposits, but lack of sufficient data renders it impossible at this time to state the conditions under which the precipitation of the barium sulphate took place and the agents involved.

VIII. Methods of Mining.

Mining of barite in Virginia is surface-work, with no deep mining in any part of the State. The greatest depth yet reached in any of the mines is 160 ft., in a shaft recently operated at the Hewitt mine, in Campbell county. In the crystalline area east of the Blue Ridge the ore is won by vertical timbered shafts and drifts which follow the direction of the ore-bodies. The machinery employed is simple, light, and inexpensive. In the limestone belt of southwest Virginia the mining is shallow and largely by open-pit work. Blasting is necessary for breaking down the ore in the fresh limestone.

IX. Preparation of the Ore.

For the removal of impurities from the better grades of merchantable ore, washing, bleaching, and grinding, and occasionally jigging, are the only necessary operations. The common impurities in the best grades of ore include the iron oxide and manganese oxide, limestone, clay, and sand. The ore in the limestone in the Valley region of southwest Virginia is both jigged and washed before bleaching and grinding. The preparation of barite has been described by Higgins.[2]

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[2] Barytes and Its Preparation for the Market, *Engineering News*, vol. liii., p. 196 (1905); Bleaching Barytes, *Engineering and Mining Journal*, vol. lxxix., p. 465 (1905).

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X. Barite-Mills.

The barite mined in Virginia is prepared for market at the following plants: The plant at Lynchburg, owned by Nulsen, Klein & Krausse; the plant at Honaker, owned by the Clinch Valley Barytes Company; the plant at Richlands, owned by the Pittsburg Barytes & Milling Corporation; and the plant at Bristol, owned by John T. Williams & Sons.

All except the Lynchburg plant are situated in southwest Virginia. These mills are modern, of large capacity, and fully equipped with machinery.

XI. Conclusions.

From the description detailed above of barite occurrence in the Virginia areas the following general facts are deducible:

(1) With two exceptions, the barite-deposits are associated with limestone or its residual decay. These exceptions show the occurrence of the barite in crystalline siliceous rocks more or less remote from limestone masses.

(2) The occurrence of the barite in the limestone is partly as a replacement, and partly as vein-like masses filling fractures; and in the residual clays as loose nodular masses irregularly assembled and of different sizes and shapes. In each of these occurrences the barite is crystalline in texture, and is the result of solution and deposition.

(3) The barite and associated minerals suggest deposition from reasonably-shallow circulations. The barite is believed to have been largely, if not entirely, derived, in most cases, from the rocks in which the concentrations are now found.

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The Promontorio Silver-Mine, Durango, Mexico.

BY FRANCIS CHURCH LINCOLN,* NEW YORK, N. Y.

(Toronto Meeting, July, 1907.)

I. Situation and Surroundings.

The Promontorio mine is situated at the northern end of the Sierra San Francisco de Coneto, in the town of Promontorio, Partido of El Oro, State of Durango, Mexico. As shown in the sketch-map, Fig. 1, the nearest railroad station is Chinacates on the Mexican International Railroad, 82 miles north of the city of Durango. The mine is 16 miles north of the station by air-line. It is reached by means of a good wagon-road which first crosses the Guatimapé Plain, passing the ranches of San Antonio and San Julian, to Estacion—a distance of about 15.5 miles—and then entering the mountains crosses the Sierra to Promontorio—a further distance of 11.5 miles—making in all a distance of 27 miles by road from railroad to mine.

Promontorio is just beyond the summit of the range, at an elevation of about 8,000 ft. above sea-level, or 1,350 ft. above the Chinacates station. The Castillo de San Francisco, the highest peak in the Sierra, has an altitude of 10,000 ft., and the Promontorio road crosses the summit of the ridge at an elevation of 9,000 ft. by a pass just below this peak known as the Puerto del Almagre. The Promontorio mill is situated at Santa Inés, 2 miles by tram or 3 miles by wagon-road to the north of Promontorio and about 600 ft. lower.

The Sierra San Francisco de Coneto decreases in height towards the north and west till it comes to an end at the Melchor Arroyo on the ranches of Melchor and Ramos. Beyond this *arroyo* another range, the Sierra de la Candela, begins. In the vicinity of the Promontorio mine the hills are very rugged and the surface is furrowed by ravines which contain running streams during the rainy season—July, August and

* Fellow in Geology, Columbia University, New York, N. Y.

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September—but are dry throughout the rest of the year. Fig. 2 is a view of the Promontorio mine, looking east, showing the

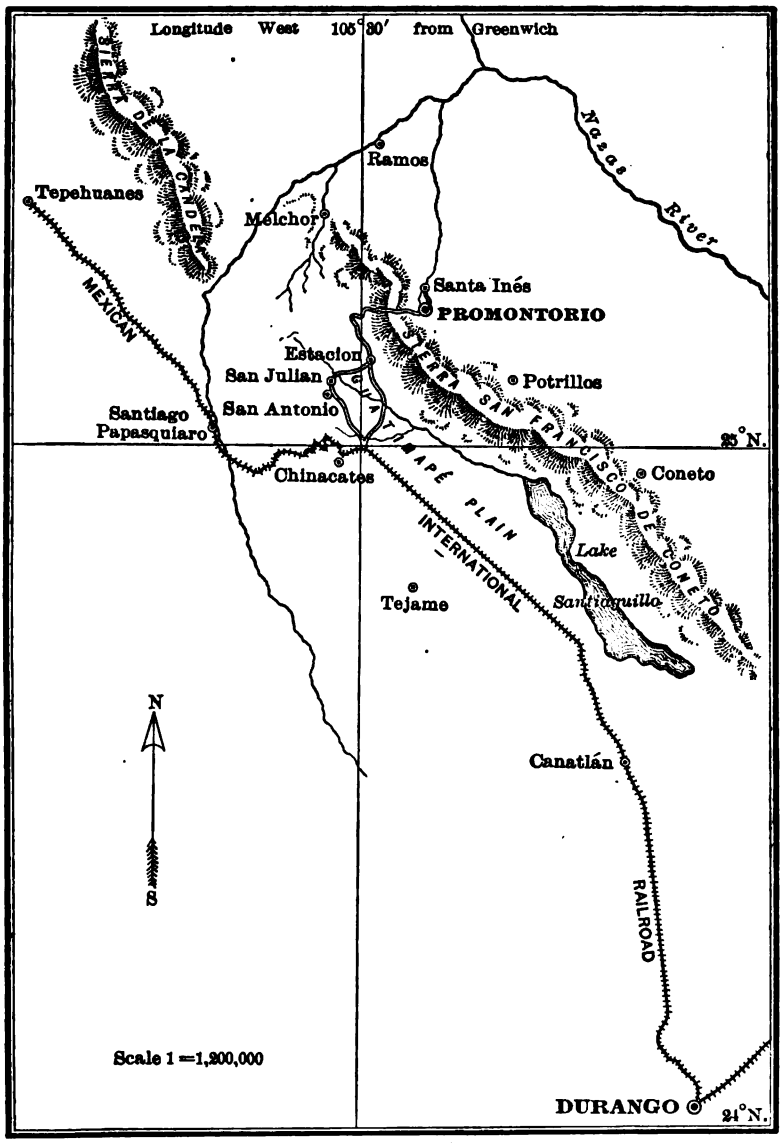

FIG. 1.—MAP OF PART OF DURANGO, MEXICO.

Refugio shaft-house in the ravine, the San Joaquin dump on the hill, the administration buildings on the left, and the power-

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house on the right. Fig. 3 is another view, looking northward. The nearest permanent flowing water is the Melchor Arroyo, a good-sized stream which might be utilized for power. A considerable number of large pine-trees are still standing on the slopes of the hills immediately adjacent to the mine, while a short distance away, oaks and other small trees are fairly abundant as well, promising a readily available supply of mine-timbers and fire-wood for a long time to come.

II. The Country-Rock.

The country-rock of Promontorio is a rhyolite-porphyry. The ground-mass is glassy, showing flow-lines, and the phenocrysts are large quartzes and orthoclases, together with smaller and somewhat altered hornblendes. Small grains of magnetite are rather plentiful, and a much lesser number of little crystals of pyrite are to be seen. The rock is, therefore, a typical rhyolite-porphyry, and undoubtedly belongs to the Tertiary rhyolites so common throughout the Sierra Madre of Mexico.

Of special interest are the inclusions. The principal ones are small, dark, angular fragments, less than an inch in diameter, which are distributed rather plentifully in some parts of the porphyry. Microscopic examination shows these inclusions to be fragments of andesite and dacite, which were evidently broken from older rocks and brought up from a depth by the rhyolite-porphyry. Occasional rounded pieces of binary granite, from several inches to a foot in diameter, are also to be observed in the porphyry. These may have been broken off from underlying-rock like the andesite and dacite; or they may be simply segregations, since they are composed of quartz and orthoclase, the commonest phenocrysts of the porphyry. According to Ordoñez,[1] the succession of Tertiary eruptives in Mexico is similar in all districts, and may be summarized as follows:

Group No. 6, Basalts—Andesitic basalts.
Group No. 5, Dacites—Andesites.
Group No. 4, Rhyolites.
Group No. 3, Andesites—Dacites.
Group No. 2, Diorites—Diabases.
Group No. 1, Granites—Granulites.

[1] Las Rhyolitas de Mexico, *Boletín del Instituto Geológico de Mexico*, No. 14, p. 66 (1900).

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If this order is correct, the Promontorio rhyolite-porphyry belongs to group No. 4. The dark inclusions in this porphyry

FIG. 2.—THE PROMONTORIO MINE, LOOKING EAST.

show that group No. 3 is represented in depth by both andesite and dacite; the binary granite, if not a segregation, is representative of group No. 1; but inclusions indicating the existence of group No. 2 have not been discovered.

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The Promontorio country-rock has a well-developed joint structure. It is split into sheets which range from less than

FIG. 3.—THE PROMONTORIO MINE, LOOKING NORTH.

an inch to more than a foot in thickness and are of considerable length and breadth. The strike of these sheets is about N. 20° W. (magnetic), and their dip nearly vertical but inclining slightly towards the northeast.

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The only other rock in the vicinity of Promontorio is a rhyolite which caps the range. At Coneto, 28 miles SE., there is an outcrop of andesite of similar character to that of the inclusions in the porphyry.

III. The Vein.

The Promontorio vein strikes through the rhyolite-porphyry N. 55° W. (magnetic). Its dip is vertical at the surface, inclining towards the SW. in depth. Thus it cuts the joint-planes of the porphyry at an acute angle in both strike and dip. The Promontorio vein proper has been followed beneath ground for a horizontal distance of 2,660 ft., but it cannot be traced so far upon the surface because of the covering of soil. Below ground the vein still continues towards the SE., while to the NW. it forks, and the West vein, proceeding from the hanging-wall of the Promontorio vein proper, has been followed an additional distance of 968 ft. The West vein can be traced on the surface much farther, and is probably identical with the La Luz vein tunneled in a prospect far to the NW. of Promontorio. To the SE. no vein has been discovered which can be identified with the Promontorio vein. Either the vein has pinched out rapidly in that direction, or, as seems much more likely, the vein is older than the rhyolite which makes its appearance to the SE. and has been covered by it.

There are no parallel veins near the Promontorio, but there are numerous cross-courses, all of which fault the NW. part of the Promontorio vein toward the SW., the horizontal displacements varying from 1 to 13 ft. These cross-courses are in some instances simple faults, while in others they have become mineralized and constitute veins. The most noteworthy example is the Veta Dolores, which strikes N. 42° E. (magnetic) and can be traced as a well-defined quartz-vein all the way from Santa Inés to its junction with the Promontorio vein—2 miles—and for some distance beyond. Another important cross-course is known as the Veta Atravesada. It is much less marked than the Veta Dolores, strikes N. 28° E. (magnetic) and is possibly identical with the Los Naufragos vein which has been explored in a prospect to the SW. of Promontorio. All the cross-courses dip steeply to the SE., which makes their

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junctions with the Promontorio vein pitch steeply in this direction also.

IV. The Ore.

The Promontorio vein is frequently divisible into three distinct parts: 1, a hanging-wall portion of vein-matter; 2, an intermediate portion of more or less altered country-rock, and 3, a foot-wall portion of vein-matter. The principal value of the ore is in silver, and sometimes one, sometimes another, portion of the vein is richest. In general, however, either the foot- or the hanging-wall portion contains the most silver, and the intermediate portion is most likely to be ore when both foot- and hanging-wall portions are rich.

Thus the Promontorio ore consists of vein-matter and mineralized country-rock. The characteristics of the fresh country-rock have already been described. It has been mineralized in two ways: 1, by silicification and impregnation with small scattered grains of the same sulphides as are found in the vein-matter; and 2, by the precipitation of secondary minerals in joint-cracks and decomposed spots.

The common primary vein-minerals are quartz, galena, and sphalerite, less pyrite, a very little chalcopyrite, and minute quantities of bornite, chalcocite, and covellite. The rare primary vein-minerals are tetrahedrite, chalcocite, argentite, and native gold. The oxidized vein-filling consists of quartz, kaolin, hematite, wad, and limonite, with occasional films of malachite and linarite and remains of the sulphides. The minerals which have contributed to the secondary enrichment are native silver, chalcocite, and a little chalcopyrite. The native silver does not contain even a trace of gold. Secondary enrichments occur both in oxidized portions of the vein and in the country-rock of the walls and horses.

Considering the frequency with which free silver is still encountered in the Promontorio mine, the rather even grade of the ore is somewhat surprising. In a systematic sampling of the mine, the highest result obtained was 263.6 oz. of silver per ton, and only 12 out of 1,059 assays exceeded 150 oz. per ton. The ratio by weight of gold to silver in the shipping-ore varies from 2 : 1,000 to 3 : 1,000 parts. The presence of small amounts of copper-minerals always indicates a high silver-content, but in all cases where neither copper-minerals nor native

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silver can be identified in the vein-filling, assays are necessary to distinguish between ore and waste.

When the assays are plotted upon the mine-map, it is seen that the ore is arranged in shoots which, like the junctions between the Promontorio vein and cross-courses, all pitch steeply toward the SE. These shoots sometimes parallel the faults, while in other cases they are cut by the faults or occur in unfaulted parts of the vein. They are usually long and narrow, extending from 15 to 100 ft. along a level and cutting many levels on their pitch. After continuing downwards for a number of levels, shoots sometimes pinch out. New shoots may come in along the line of the old ones, or make their appearance in intermediate positions.

The shoots are either primary or secondary, both pitching in the same general direction. The primary shoots are distinguishable by their comparatively high content of sulphides, by their lack of secondary minerals, and by their habit of being cut off by faults unless occurring in unfaulted parts of the vein. On the other hand, the secondary shoots are recognizable by their low content of sulphides, by the presence in their richer portions of the secondary minerals, native silver, chalcocite, and chalcopyrite, and by their tendency to follow closely well-defined faults. The primary ore-shoots are dominant in the lower levels, the secondary in the upper. The secondary ore-shoots reach their maximum development in the neighborhood of the fourth level, where the Veta Dolores shoot extends horizontally for 460 ft. and the Veta Atravesada shoot for 456 ft.

None of the cross-veins have developed ore at a distance from the Promontorio vein, although small amounts of ore have been taken from some of them near their junctions with the Promontorio vein. Considerable prospecting has been carried on, but no other mine has been discovered in the district.

V. The Relations of the Primary Vein-Minerals.

Polished surfaces of the sulphide ore were prepared and examined microscopically under the direction and after the methods[2] of Dr. William Campbell, for whose assistance I de-

[2] *Economic Geology*, vol. i., pp. 751 to 756 (1906).

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sire to express my thanks. The specimens consisted, for the most part, of three minerals, galena, sphalerite, and quartz, whose relations are illustrated in Fig. 4. This shows that the sphalerite was formed first. It was fissured, and subsequently quartz was deposited in the fissures, and also in crystals on the exterior. Finally came galena, filling the vugs in the

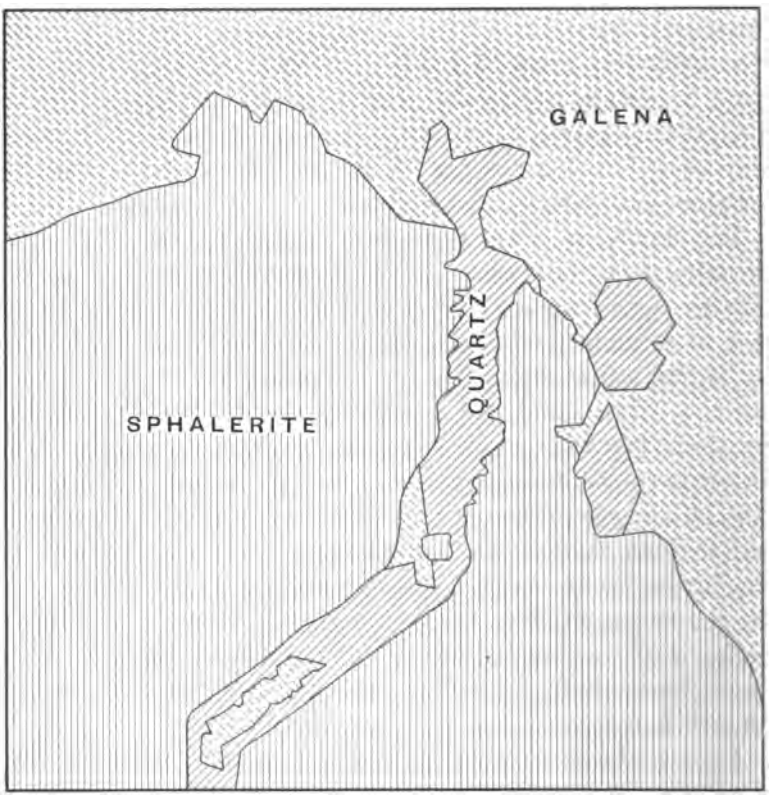

FIG. 4.—LOW-GRADE SULPHIDE ORE. (Magnified 80 times). RELATIONS OF SPHALERITE, QUARTZ, AND GALENA.

quartz-vein and molding itself about the previously deposited sphalerite and quartz.

Pyrite is an important component of the sulphide ore, though by no means so plentiful as sphalerite, galena, or quartz. The relation of pyrite to these minerals is clearly indicated in Fig. 5. Pyrite is there shown to be incrusted with sphalerite, which is, in turn, incrusted with quartz-crystals, while galena occupies

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all the remaining space, just as it does in Fig. 4. Pyrite, therefore, clearly antedates sphalerite, and was the first vein-mineral to be formed.

Chalcopyrite is a much rarer constituent of the primary ore. Its usual mode of occurrence, illustrated in Fig. 6, is in grains attached to the sphalerite and surrounded by galena. In this illustration one grain is shown surrounded by bornite and another is intimately associated with covellite. In other specimens chalcocite has been found in similar relations with chal-

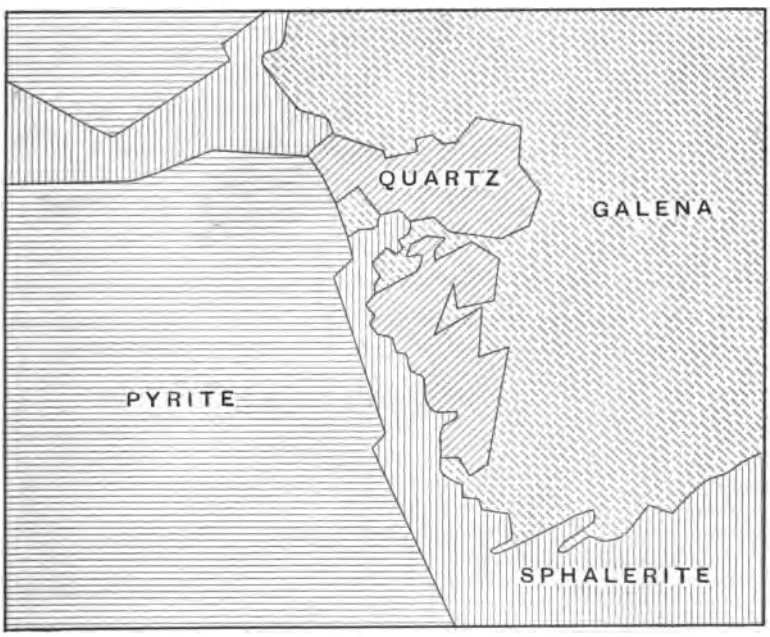

FIG. 5.—LOW-GRADE SULPHIDE ORE. (Magnified 80 times). RELATIONS OF PYRITE, SPHALERITE, QUARTZ, AND GALENA.

copyrite. These associations indicate that the minute amounts of bornite, chalcocite, and covellite found in the ore are probably secondary and derived from chalcopyrite. Chalcopyrite has not been found contiguous to quartz, but since it is entirely absent from quartz-veins in sphalerite and present in galena-veins in this same mineral, it is reasonable to infer that it is later than the quartz. The secondary copper-minerals associated with grains of pyrite are illustrated in Fig. 7, which shows that the bornite is younger than the quartz and older than the chalcocite.

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The order of succession of the minerals in the primary ore of Promontorio is therefore: 1, pyrite; 2, sphalerite; (period of crushing); 3, quartz; 4, chalcopyrite; 5, galena.

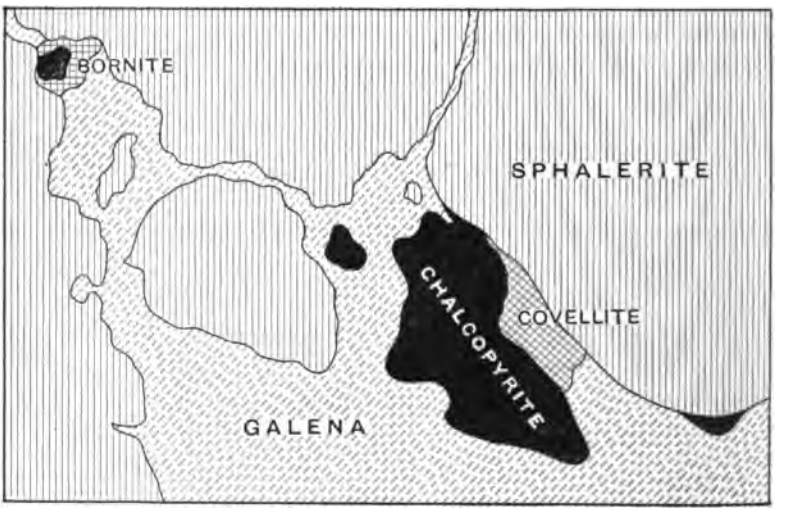

FIG. 6.—MEDIUM-GRADE SULPHIDE ORE. (Magnified 80 times). RELATIONS OF SPHALERITE, CHALCOPYRITE, COVELLITE, BORNITE, AND GALENA.

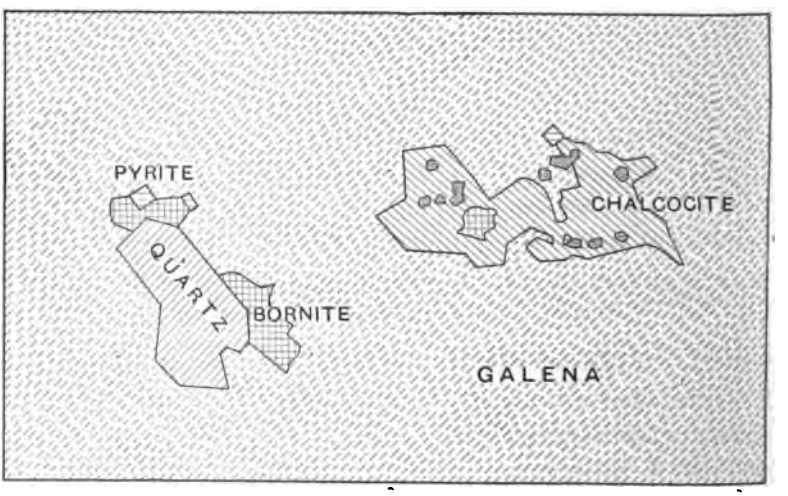

FIG. 7.—MEDIUM-GRADE SULPHIDE ORE. (Magnified 80 times). RELATIONS OF PYRITE, QUARTZ, BORNITE, CHALCOCITE, AND GALENA.

Rich primary minerals are exceedingly rare at the Promontorio mine. They play no recognized part in the mine's pro-

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duction, although they are probably present in minute amounts in the ordinary primary ore.

A specimen of rich ore obtained from the fourth level is apparently of primary origin. It consists of numerous metallic gray stringers and veinlets in milky white quartz. Polished surfaces, examined under the microscope, show the cavity-fillings to consist of tetrahedrite, chalcocite, argentite, and native gold, arranged as illustrated by Figs. 8 and 9. (These minerals were identified by methods which I developed and shall soon publish.)

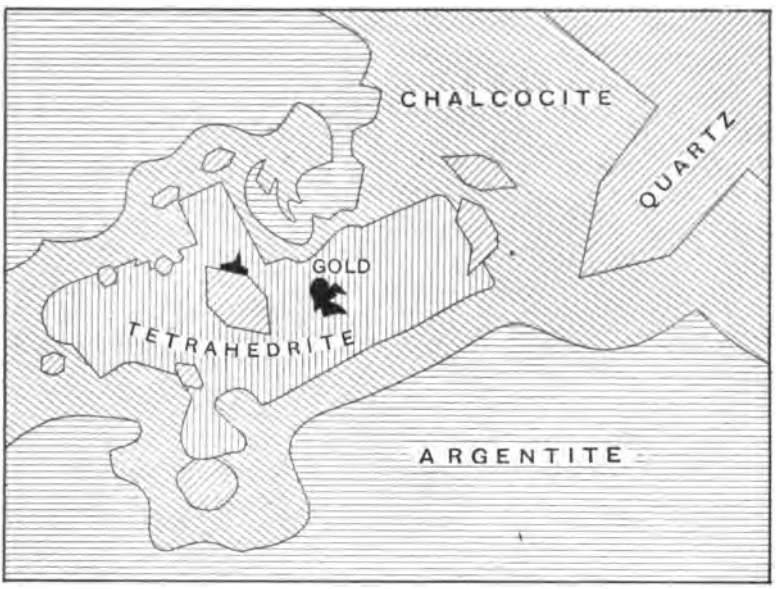

FIG. 8.—HIGH-GRADE SULPHIDE ORE. (Magnified 200 times). RELATIONS OF QUARTZ, GOLD, TETRAHEDRITE, CHALCOCITE, AND ARGENTITE.

Quartz was deposited, then shattered, and native gold deposited upon it in small isolated crystals. Tetrahedrite came next, forming detached crystals also, which occasionally surrounded gold. A crust of chalcocite then formed over all, and what gold had not already been involved in tetrahedrite became surrounded by chalcocite. This is indicated by Fig. 9. The section of veinlet there represented was completely filled with chalcocite, but in most instances the central parts of the crevices were left open and were filled later with argentite, as shown in Fig. 8.

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The order of succession in this rich piece of ore was therefore: 1, quartz; (period of crushing); 2, native gold; 3, tetrahedrite; 4, chalcocite; 5, argentite.

It will be noticed that the period of crushing came just after the deposition of quartz instead of just before, as in the ordinary primary ore. The difference in conditions thus clearly indicated probably caused the deposition of particularly rich ore at the point whence this specimen was taken.

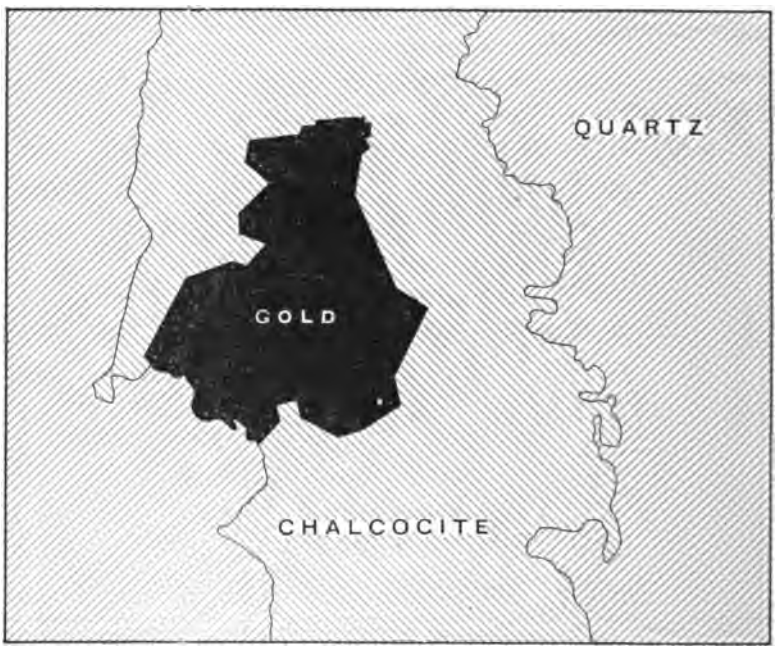

FIG. 9.—HIGH-GRADE SULPHIDE ORE. (Magnified 200 *times*). RELATIONS OF QUARTZ, GOLD, AND CHALCOCITE.

VI. GEOLOGICAL HISTORY OF THE PROMONTORIO DISTRICT.

The Promontorio rhyolite-porphyry was extruded during Tertiary time through underlying andesite, dacite and, perhaps, binary granite. Pressure during cooling developed sheet-jointing.

After the consolidation of the porphyry a disturbance took place which resulted in the formation of an extensive NW.–SE. fault, through which heated ore-bearing solutions rose. This disturbance was probably caused by an intrusion of molten igneous rock that faulted the overlying layers and gave forth

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plutonic emanations, which, ascending through the fault, deposited the primary ore of the Promontorio vein.

The conditions were such that first pyrite, then sphalerite was deposited. A slight rearrangement along the fault led to the shattering of some of the pyrite and sphalerite, and at the same time brought about a change of conditions, so that quartz and, soon after, chalcopyrite were formed. Finally, galena was deposited in all remaining cavities, and the deposition of primary minerals came to an end.

After the mineralization of the Promontorio fault another upheaval took place, which was probably contemporaneous with the extrusion of the rhyolite that caps the Sierra San Francisco. This caused the formation of a series of NE.–SW. faults, several of which intersected the Promontorio vein. These faults were slightly mineralized, perhaps by lateral secretion from the porphyry, but certainly in a different manner from the Promontorio fault.

The district was covered by rhyolite for a while, but, when this was eroded, the only partly filled cross-faults formed convenient channels by which surface-waters were tapped off, and, entering the Promontorio vein in the neighborhood of these cross-faults, rearranged its contents. Rich deposits of secondary minerals were thus formed in the vicinity of the cross-courses.

VII. Mining.

The Promontorio mine was discovered by Joaquin Contreras in 1880, and purchased by its present owner, the Negociacion Minera de Promontorio, S. A., in 1887. It is at present the only real mine in the whole Sierra San Francisco de Coneto district. It should be noted, however, that the famous Potrillos tin-deposits are situated in these mountains, and that a system of silver-gold veins at Coneto has been worked intermittently for a long period of time.

When mining began at Promontorio, it was necessary to freight the ore 260 miles to the railroad at Fresnillo. At that time, shipping-ore had to contain at least 240 oz. of silver per ton. In 1892 the railroad reached the city of Durango, making it profitable to ship 90-oz. ore; and when, in 1900, the branch line was opened to Chinacates, 60-oz. silver-ore could be shipped at a profit.

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The mine was first opened by the San Joaquin shaft, on a hill-side (see Fig. 2), and later by the Cinco Señores workings, still further up the hill. The present main working-shaft, the Refugio, is in the ravine beside the hill upon which the older openings are situated. The shaft-house is visible in Figs. 2 and 3.

This shaft has been sunk to a depth of 675 ft., of which the first 280 ft. are in the vein, and 14 levels run from it. Levels 1 to 9 are 40 ft. apart; levels 9 to 14 are 71 ft. apart. Level 1 is a tunnel which starts at the mouth of the shaft and passes into the hill to the SE. where it connects with the early workings mentioned above. This level has the furthest extent towards the SE. of any. The other levels are all drifts run in both directions on the vein for longer or shorter distances. Level 9 extends furthest towards the NW., passing the fork in the Promontorio vein into the West vein and finally connecting with another shaft, the Santa Maria, at a depth of 250 ft. From the SE. face of the first level to the NW. face of the ninth level is a horizontal distance of 3,628 ft. A cross-cut from a point in the Santa Maria ravine a short distance below the Santa Maria shaft intersects the fourth level and converts it into a tunnel.

When I visited the mine in the fall of 1906, the method of operation was as follows: All the material mined was trammed out to a "*patio*" or sorting-yard by means of the cross-cut from the fourth level. Rock mined above the fourth level was dropped to it, and rock mined below was raised to it by the hoist at the Refugio shaft. On the *patio* the produce of the mine was sorted to "shipping" ore, running 60 oz. of silver per ton, and better. The rejected material was thrown on the milling-dump when it ran better than 20 oz., and on the waste-dump when worse. Since I left Promontorio, a mill has been started, and it is probable that ore of the grade formerly shipped is now being milled together with the lower-grade material.

The power-house, situated in a small *arroyo*, across from the main shaft (see Fig. 3), contains six tubular boilers, which burn wood and supply steam to the hoist and compressor, and to engines operating generators. The electricity generated supplies power at the *patio*, illuminates electric lamps, and operates the

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pumps. A 120-kw. alternating-current generator supplies the power to four 3-in. Worthington electric pumps. In the dry season the pumps are worked for 8 hr. a day, and raise about 96,000 gal. of water to the fourth level daily. Most of this water comes from the fourteenth level, a distance of 555 ft. In the rainy season, it is necessary to run the pumps 14 hr. a day, thus raising 168,000 gal. per day.

Fire-wood delivered at Promontorio costs $2.50 per cord and lumber $17.50 per 1,000 ft. The wages paid per day are:

Miners,	$0.75
Laborers,	0.375 to $0.75
Machine Drillers,	1.25
Shift Bosses,	1.00 to 1.25
Timbermen and Carpenters,	0.875
Blacksmiths,	1.25
Hoistmen,	1.25
Engineers,	1.00
Firemen,	0.625

Labor is, on the whole, plentiful. Some difficulty is experienced in keeping sufficient men at the mine during the periods of sowing and harvesting of crops, for the Durango laborer prefers farming to mining. Another peculiar condition has to be met in cold weather, when all hands want to work on night-shift because their cabins are too cold to sleep in at night, but are warmed to a more comfortable temperature by the sun during the day. With the exercise of a little tact there should be no difficulty in obtaining all the labor desired.

VIII. Milling.

Milling operations did not prove a great success at Promontorio in the early days of the mine. One brand-new mill, which had never turned a wheel, was completely destroyed by the collapse of a dam. Another mill was erected later, but failed to give good extractions, as the tailings-dumps bear witness.

Mr. Gordon Wilson carried on a long series of milling-tests upon Promontorio ore, and came to the conclusion that the best results could be obtained by concentration, followed by sliming and cyanidation of the tails. His experiments indicated that 45 per cent. of the values could be extracted by concentration in 10 per cent. of the weight, and that, of the remaining 55 per cent., 89 per cent. of the silver, and practically all the gold,

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could be extracted by sliming and cyaniding for a period of 10 days. This would mean a total extraction of 94 per cent. Mr. Wilson, therefore, constructed a 50-metric-ton concentrating and cyaniding mill at Santa Inés, which has now been in operation for several months, and, I am informed, is performing good work.

The dam which supplies water for this mill is situated midway between Promontorio and Santa Inés. It is 100 ft. high, and was constructed at a cost of $50,000.

IX. Production.

The smelter-returns on shipments from Promontorio are reported to have been about $5,000,000. Unfortunately, no records of the quantities of gold and silver produced were kept during the early bonanza days of the mine. From Dec. 5, 1896, to Aug. 18, 1906, there were produced and sold 5,689,618 oz. of silver and 15,857.4 oz. of gold. During this period of recorded production the Promontorio mine lost its position as "one of the largest producers of silver in Mexico,"[3] and became one of the many minor producers. At present there is but little shipping-ore in sight in the mine, but there are large reserves of good milling-ore, and, with new and effective milling methods in full operation, we may soon expect to see the Promontorio mine make its way to the front once more.

[3] Ingalls, *Trans.*, xxv., 149 (1895).

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The Evergreen Copper-Deposit, Colorado.

BY ETIENNE A. RITTER, COLORADO SPRINGS, COLO.

(Toronto Meeting, July, 1907.)

INTRODUCTION.

THE Evergreen mine, located at Apex, in the northern part of Gilpin county, Colorado, has opened a very peculiar and interesting copper-deposit, in which both bornite and chalcopyrite occur as rock-minerals.

The country-rocks are crystalline schists cut by dikes of granulite and of pegmatite. At the Evergreen property they have been cut by a dike of a new kind of eruptive rock, to which I have given the name "Evergreenite." The rock is composed of quartz, alkali feldspars, orthoclase and albite (often interlocked as microperthite), with augite of the ægirine variety and long needles of enstatite and diallage. For reasons stated below, I believe the dike to be of Tertiary age. It is easily traced on the surface of the ground, from a point in the bottom of Pine creek, about a mile below Apex, to the top of Nevada hill, 1,500 ft. beyond and 600 ft. above it. (See Fig. 1.) It varies from 3 to 12 ft. in width, and is bounded on both sides by contact-zones about 60 ft. wide, in which the crystalline schists have been changed into pseudo-quartzites or pseudo-gneisses, according to the injection of an excess of quartz or of feldspars through the special layer of the schists. The most interesting feature is the presence of the pyroxenes, which are always found, though in very variable proportions from place to place, in the layers of these metamorphosed crystalline schists.

The dike has torn from its walls a large number of fragments of the wall-rock. It must have been in a viscous state when that happened, because these fragments of the wall-rock do not show any sign of resorption or of assimilation by the magma. In places they are so distinct and so numerous that the rock looks to the naked eye like a pseudo-breccia, of which, never-

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theless, the true nature is easily recognized in the microscopic slides. The most striking feature is the great difference in freshness between the eruptive rock and the inclosed fragments of the wall-rock. The former is marvelously fresh. It does not show any decomposition-products. The feldspars of the inclosed pieces of wall-rock, on the contrary, are completely changed into sericite; but the biotite is fresh yet. There has been a strong sericitization, but no propylitization.

The dike-rock shows microscopic structures, which, in many places, blend into one another, but may be quite distinct at other places. These structures are granulitic, with a tendency to become micropegmatitic and porphyritic.

A large proportion of the ore is found in the dike itself, in which chalcopyrite and bornite appear as rock-minerals, to the same degree as the quartz, the feldspars or the pyroxenes. Fig. 2 clearly shows this occurrence. These copper sulphides must have come up dissolved in the magma at the time of its ascent to form the dike, and must have crystallized together with the silicates of the eruptive rock. It is a striking example of the deposition by sublimation, under pneumatolytic action, so ably and completely worked out by Elie de Beaumont, Henry Sainte-Claire Deville and Daubrée, towards the middle of the last century. Since then it has been shown that deposits of this kind are the rare exceptions. But they are very important in throwing a great deal of light on the way the majority of the ores can come from great depths to within a distance of the surface small enough to allow the circulating waters to dissolve them and to bring them up in the veins.

I do not know of any other copper-deposit in which the genesis of the ore, as a rock-mineral, segregated in the cooling of the eruptive magma, is so plain and can be so well illustrated by the study of microscopic slides. The proposition that bornite is a rock-mineral is not new. It has been already proved by Prof. J. F. Kemp[1] and Jules Catherinet[2] in a pegmatite dike in the Similkameen district, B. C.

The view that some copper-deposits may have been formed by segregation from a magma has always been held in Europe,

[1] *Trans.*, xxxi., 182 (1901).

[2] *Engineering and Mining Journal*, vol. lxxix., p. 125 (1905).

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FIG. 1.—SHAFT-HOUSE OF THE EVERGREEN MINE. THE DIKE FORMS THE CREST OF THE RIDGE.

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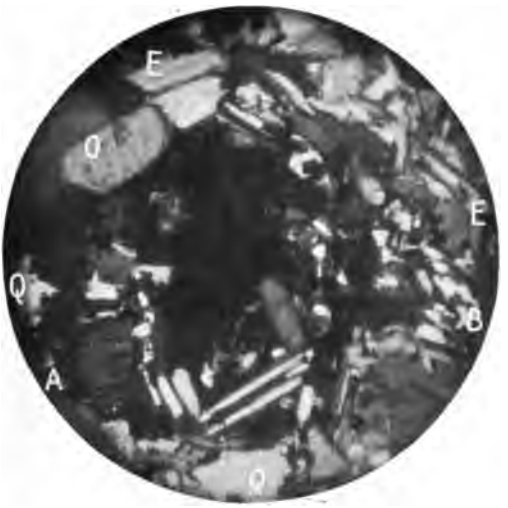

FIG. 3.—The opaque areas, B, bornite; dark-shaded areas, A, augite-ægirine; long white needles, E, enstatite or diallage; light areas with few cracks, O, an alkali feldspar, orthoclase, albite or microperthite; other light areas, Q, quartz. (Without analyzer. Magnified 71 diameters.)

FIG. 4.—"Evergreenite" with porphyritic structure. The large crystal in the center is quartz, containing inclusions of bornite. (With analyzer. Magnified 71 diameters.)

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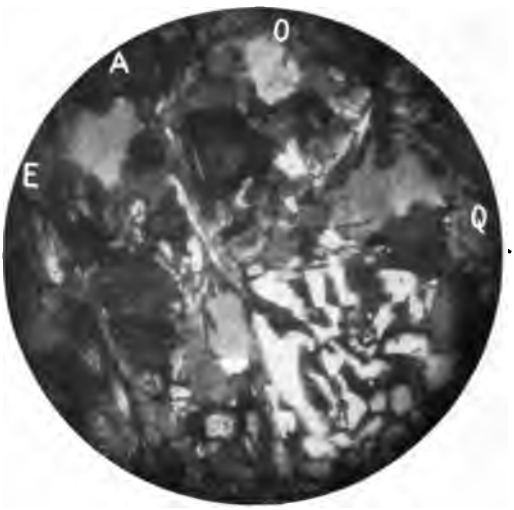

FIG. 5.—"Evergreenite," intermediate between microgranitic and porphyritic types, but without bornite. Note a patch of granophyre. (With analyzer. Magnified 71 diameters.)

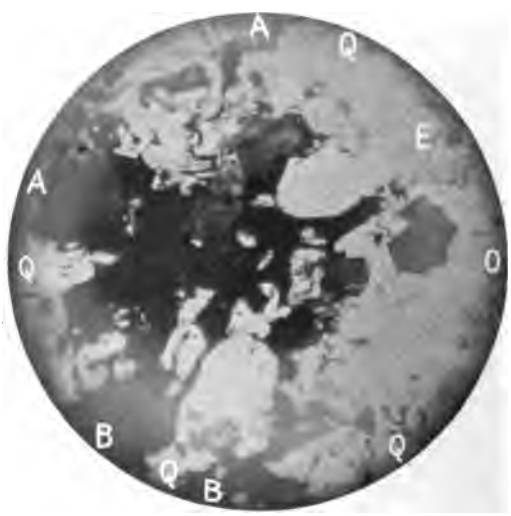

FIG. 6.—Bornite surrounding crystals of quartz, enstatite and augite-ægirine, and itself surrounded by the same minerals and orthoclase, with inclusions of bornite in the quartz and the augite. (Without analyzer. Magnified 71 diameters.)

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and has had as advocates Fuchs and de Launay in France, J. H. L. Vogt in Norway, A. Schenk in Germany and B. Lotti in Italy.

Among the conspicuous examples of such deposits are the copper-deposits of the Banat on the boundary of Hungary and of Roumania; the celebrated ore-deposit of Monte Catini,[3] and others less important, found in Tuscany;[4] and the ore-deposits of Ookiep,[5] in the Kleinnamaland, in South Africa; also some deposits in Prince of Wales Island, Alaska,[6] and Shingle Springs, California.[7]

The Petrology of the Ore-Deposit.

The Evergreen ore-deposit lies at an altitude of 10,200 ft., on the eastern slope of the Front Range. It crops out on a hill with a northerly exposure, where the snow remains on the ground for six or seven months every year, preventing the formation of an oxidized zone of any importance. The dike has taken its name from the Evergreen claim, which covers it for 1,500 ft. in length. It is opened at its lower end by a tunnel about 700 ft. long, and by a shaft with three levels, at 100, 150 and 200 ft. The dike strikes nearly N–S. and dips from 65° to 75° E. It is not followed continuously by any level, but it has been cut by the levels, and in several places followed for some distance.

In the dike, as exposed by the mine-workings, a fair amount of the original ore has been left unaltered by subsequent action. This is due to the tightness of the eruptive rock. The metamorphosed zones of the gneisses and altered crystalline schists, on both sides of the dike, have been much less impervious to the circulation of vadose waters; and a secondary enrichment can be observed in these zones.

---

[3] Fuchs et de Launay, *Traité des gîtes minéraux et métallifères*, vol. i., p. 660, and vol. ii., p. 230 et suiv.; also L. de Launay, Contribution à l' Étude des Gîtes Métallifères, *Annales des Mines*, Ninth Series, vol. xii., pp. 119 to 228 (1897).

[4] B. Lotti, La miniera di Montecatini, *Bollettino del Reale Comitato Geologico d'Italia*, vol. xv. (1884).

[5] A. Schenck, *Zeitschrift der Deutschen Geologischen Gesellschaft*, vol. liii., p. 64 (Verhandlungen der Gesellschaft) (1901).

[6] W. W. Rush. A Curious Occurrence of Copper, *Mining and Scientific Press*, vol. xciii., p. 624 (1906).

[7] C. Y. Knight. A Curious Occurrence of Copper, *Mining and Scientific Press*, vol. xciv., p. 242 (1907).

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The eruptive rock composing the dike is of a new type, as illustrated in Figs. 3 and 4. It shows a wide range in structure, varying from microgranitic to porphyritic, with all the intermediate stages, as shown in Fig. 5.

The microgranitic part is more acid than the porphyritic; the crystalline schists of the metamorphosed zone have been transformed by an injection of the magma of the more acid type.

In the acid type, the phenocrysts are those of the alkali feldspars, mostly microperthite. They are very fresh, and contain some microscopic needles of enstatite and of diallage lined up in their cleavages. The crystals are seldom broken, and when that happens, the narrow cracks are filled by very small flakes of sericite.

The older quartz occurs in large and irregular grains, more broken than the feldspar crystals; the several grains forming a large patch do not die out simultaneously.

The few phenocrysts of ægirine show strong pleochroism. They are surrounded by belts of granulitic quartz, intergrown with numerous tabular crystals of ægirine, and needles of enstatite and of diallage showing all kinds of orientation. There are but few sections of the sphene, and fewer still of magnetite. A typical micropegmatitic intergrowth of quartz and feldspar occurs in small patches, giving arborescent figures in polarized light. The specimens of this class are richer in feldspar and poorer in pyroxene than those of porphyritic texture.

In the latter, it seems that the segregation of the magma is more advanced, and there is a consequent decrease in the amount of the feldspars. These specimens show a part (more acid) specially rich in quartz, and another part (more basic) characterized by the abundance of needles of enstatite and of diallage, with sharp crystalline forms, lined up almost well enough to indicate a direction of flowage. There are very few phenocrysts of intra-telluric formation surrounded by a groundmass of smaller ones. On the contrary, there seems to have been only one period of crystallization; and all the crystals are of about the same size.

The augite is in smaller amount, while the needles of enstatite and of diallage are more numerous than in the other microgranitic type. There is very little magnetite or sphene. The

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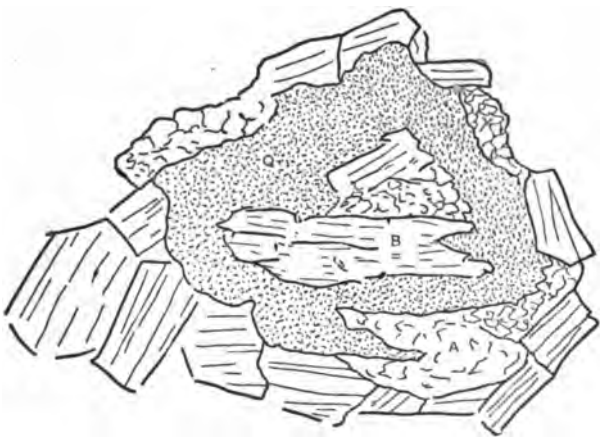
rock is also very fresh in this type, and shows no decomposition-products.

This freshness of the rock is in striking contrast with the altered condition of the country-rock around it and is a strong argument in favor of a Tertiary age for the dike. While the time of its ascent can only be guessed at, we may not unreasonably associate it with that of so many Tertiary volcanic rocks, which form a long N–S. belt on the eastern slope of the Front Range.

The included fragments torn from the wall-rock are more numerous in the microgranitic than in the porphyritic part of the dike. They consist of crystalline schists, rich in biotite



A, Augite. B, Biotite. L, Leucoxene. Q, Quartz.

FIG. 7.—WALL-ROCK SURROUNDED BY AUGITE, IN A SLIDE OF ERUPTIVE ROCK. (Magnified 40 times.)

and in magnetite, with feldspars always thoroughly sericitized. The biotite is not altered. These inclusions are of all sizes, from that of a phenocryst, perhaps 0.02 in. in diameter, to several cubic feet. The smallest inclusions are often surrounded by a belt of small tabular crystals of augite (Fig. 7). Their line of contact with the eruptive rock is sharp. In the few cases where the included fragment of the wall-rock has been partly resorbed by the magma of the eruptive rock, it appears as a ground-mass of microgranitic quartz, sericite, magnetite, biotite and leucoxene, without any marked texture. Cubes of pyrite or of chalcopyrite are sometimes seen in these fragments.

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The crystalline schists cut and metamorphosed by the dike show two main types of rocks. They have been changed either into pseudo-gneisses or into pseudo-quartzites. In the first case, they are made mostly of alkali feldspars (orthoclase and albite, occurring often as microperthite). These have been largely changed into sericite; there are few crystals of oligoclase and some large and broken quartz-grains. The crystals of biotite are broken, but they are rather fresh. In places they have lost their pleochroism and their interference-colors; but they have not been changed into a new mineral. More than half of the rock is made of feldspars. The sections show also a few grains of magnetite and some larger grains of pyrite, of chalcopyrite and, more seldom, of bornite. The ægirine is more abundant and occurs in crystals without sharp outlines, green, but lacking in pleochroism. It is often partly changed into leucoxene.

The quartzose injection has changed the original crystalline schists into a ground-mass very rich in quartz. The biotite is in needles, smaller and more numerous than in the pseudo-gneisses; the magnetite is more abundant and the ægirine less so. The feldspars are mostly oligoclase; their crystals are smaller and less numerous than in the gneissic type.

The most instructive sections are those showing the relations between the copper-minerals and the eruptive rock. These sections may be divided into two groups—those which show bornite only as the copper-mineral, exhibiting also very plainly the relations of the bornite as a rock-forming mineral; and those which show not only bornite, but chalcopyrite and some covellite also. This latter group comprises also the sections in which the ore is at the contact of the inclosed fragments of wall-rock with the eruptive magma. The relations between the silicates of the eruptive magma and the copper sulphides are not so easily deciphered in them; and in places the action of secondary agencies can be plainly detected.

The slides of the first group show not only that the bornite is a rock-mineral, but also that it has not crystallized at a single time. Being a sulphide instead of a silicate, it has possessed special fluidity; and, as a result, it is found as inclusions in the quartz, the feldspars (Fig. 8), the ægirine and in

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the clusters made by the needles of enstatite and of diallage, showing that it crystallized sometimes before any one of these

A, Augite. B, Bornite. E, Enstatite. Q, Quartz.

FIG. 8.—QUARTZ- AND ORTHOCLASE-CRYSTALS WITH INCLUSIONS OF BORNITE. (Magnified 40 times.)

minerals (Fig. 9). On the other hand, the patches of bornite, which contain as inclusions, at one spot or another, each one

A, Augite. B, Bornite. E, Enstatite.

FIG. 9.—AUGITE WITH INCLUSIONS OF BORNITE. (Magnified 40 times.)

of these various silicates, show that the bornite crystallized last of all (Fig. 10). A curious slide shows an intergrowth of

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crystals of enstatite, diallage and bornite, with a texture similar to that of the quartz and feldspar in "graphic granite" (Fig. 11).

A, Augite. E, Enstatite. O, Orthoclase. Q, Quartz.

FIG. 10.—AUGITE, ENSTATITE, ORTHOCLASE AND QUARTZ SURROUNDED BY BORNITE. (Magnified 30 times.)

The accompanying sketches and microphotographs show these facts better than a long description (See Figs. 2 to 12).

Only small parts of the ore-deposit exhibit secondary en-

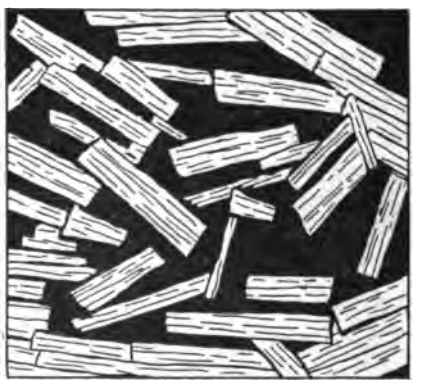

FIG. 11.—BORNITE AND NEEDLES OF ENSTATITE AND DIALLAGE INTERLOCKED, WITH THE TEXTURE OF A GRAPHIC GRANITE. (Magnified 50 times.)

richment; and these parts are always in the midst of the metamorphosed crystalline schists, at their contact with the eruptive rock of the dike.

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In these crystalline schists, the exomorphic addition of magma consists chiefly of quartz and ægirine. There has been a partial resorption of the broken mass of the schists by the granophyric quartz; and the tabular crystals of augite are scattered through that ground-mass. There is every indication that the quartz of the ground-mass is primary and that it has not been deposited by the descending waters which have changed the bornite and the chalcopyrite into covellite. The freshness of the broken crystals of augite and of biotite, which have kept their pleochroism, as well as the texture of the ground-mass of quartz, are opposed to such a secondary deposition.

The metasomatism is localized and concentrated along the channels of water-circulation, and never extends beyond a narrow margin on each side of them. Even in the slides in which the bornite has been altered into covellite, it is easy to recognize the primary characteristics of the former as a rock-mineral. The metasomatic action has failed to change the outlines of the patches of bornite, or their relations with the neighboring crystals of the eruptive rock. The relations of the covellite with the bornite and with the chalcopyrite are plainly those of a secondary mineral. They show a striking contrast with the relations of the bornite and chalcopyrite towards one another.

These two minerals occur as primary materials of the rock. They are distributed in separated patches, sometimes wide apart, sometimes adjoining one another, or following each other, along the general trend of flowage marked by the parallelism of the microlites of enstatite and diallage. But neither of them ever forms a margin to the other, following its various embayments, and neither ever penetrates the other along a line of fracture, as is always the case with the covellite. I have never been able to observe a case in which the chalcopyrite has been enriched and changed into bornite, as is often the case in copper-deposits. The secondary enrichment of the chalcopyrite, like that of the bornite, always gives covellite and limonite. I have also failed to observe any such transformation, by leaching, of bornite into chalcopyrite as has been observed by Mr. Jules Catherinet in the primary bornite of the Similkameen deposits. The bornite and the chalcopy-

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rite act towards each other like two minerals of slightly different chemical compositions which have crystallized at the same time, during the cooling of the eruptive magma, as did, for instance, the augite and the enstatite or the diallage in the "Evergreenite."

On the contrary, the secondary enrichment, transforming bornite and chalcopyrite into covellite, with formation of secondary limonite, is very plain (Fig. 12). The alteration has

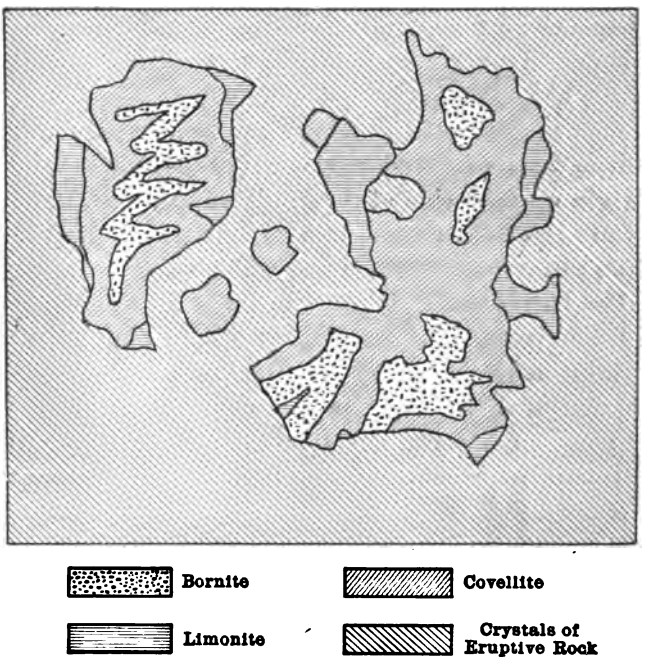

FIG. 12.—SKETCH SHOWING THE ALTERATION OF BORNITE INTO LIMONITE AND COVELLITE.

begun along the margin of the patches of these minerals and has penetrated all the crevices towards the heart of their crystals. As the rock has been but slightly compressed, its crystals are seldom broken; and this has handicapped the work of the copper-bearing solutions. Only the smallest patches of bornite have been completely altered into covellite and limonite, without leaving any trace of the primary mineral. In almost every case the covellite forms only a margin surrounding more or less completely the patch of bornite or of chalco-

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pyrite, allowing every step of the metasomatic process to be easily followed.

This secondary enrichment is the only alteration shown by the ore, and it is always accompanied by a strong alteration of the feldspars and of the neighboring augite. If the alteration of the eruptive rock near the covellite is very striking, it is also quite local and stops within a very short distance. The feldspars show little sericitization, but all their cleavages are lined with limonite. The same features can be observed as to the ægirine. That mineral has always lost its pleochroism and is often also altered more or less completely into leucoxene.

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The Geological Relations of the Scandinavian Iron-Ores.

BY PROF. HJALMAR SJÖGREN, PH.D., STOCKHOLM, SWEDEN.*

(Toronto Meeting, July, 1907.)

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* Director of the Mineralogical Department of the State Natural History Museum, Stockholm, Sweden.

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INTRODUCTION.

From a geological point of view, the iron-ores of the Scandinavian peninsula may be classified as follows:

I. *The ores of the Archæan crystalline schists.* These ores belong to the division of the Archæan formation, crystallized in the anamorphic zone and interwoven with granites; the ores occur associated with ortho- and para-gneisses, granulites and dolomized or silicified limestones.

II. *The ores of the porphyries (keratophyres)*, belonging to a division of the Archæan formation, plicated in pre-Cambrian time, but younger than the old granites.

III. *The ores of the basic eruptive rocks*, occurring as differentiations in intrusives of diabase, gabbro and norite, forming stocks, bosses or laccolites within schists of Archæan and Silurian age.

IV. *Ores occurring in the metamorphosed Cambro-Silurian schists*, chiefly in the mica-schist group, characterized by mighty beds of limestone.

V. *Contact-formations connected with acid eruptive rocks* of post-Silurian age, in the Archæan gneisses and in the Silurian limestones and argillaceous schists.

VI. *Lake- and bog-ores*, belonging to the most recent geological period.

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This classification is neither genetic nor based on the age of the ores. However, the above groups are well defined, and hardly any transitions between them can be found. There is generally no difficulty in deciding to which of the groups a certain ore-deposit is to be referred. In drawing up the present scheme a consideration of the rock or the geological unity with which the different ores are connected has furnished the chief ground of classification.

In this connection a review of the geological age of the different kinds of ore-bearing rocks—which is not always the same as the age of the ores—is of some interest.

The ores of Group I. occur in those older Archæan rocks which are interwoven with granite. With respect to their present characters, form, textures and mineral constitution, the ores of this group are younger, or, at least, no older, than the plication of the rocks.

As for the age of those rocks with which the ores of Group II. are connected, it seems not to be established with certainty, but it is evident that these rocks belong to a younger division of the Archæan. The ores frequently exhibit epigenetical characters with respect to the rocks.

The ores of Group III. are connected with rocks of highly varying age: Taberg in Småland forms a laccolite of olivine-diabase in the oldest Archæan formation; Routivare is an intrusion in the metamorphosed Cambro-Silurian schists of the Scandinavian mountain-ridge and is, consequently, younger than these, which is, probably, also the case with the ore-bearing gabbro of Lofoten and Vesteraalen. The nepheline-syenite in Alnö is post-Archæan. The ores belonging to this group are syngenetic formations.

The ores of Group VI. lie in rocks of Silurian age; but it was only in connection with, or after, the post-Silurian rock-plication that the ore-formation was finished.

The ores included in Group V. occur chiefly in the normally developed Silurian formations of the Christiania basin, as well as in Archæan gneisses, but they are genetically connected with post-Silurian eruptive rocks, and consequently of the same age as these.

The different geological ore-groups are characterized by

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mineralogical and chemical properties, which have been determinative of their technical utilization.

The lake- and bog-ores of Group VI. were, on account of their cheap exploitation and easy reducibility, the earliest raw-material of iron-manufacture in the Scandinavian countries; but their importance has been decreasing as the iron-manufacture has become a great industry. They occur chiefly in Sweden, but also in Norway.

The ores of Group I. were next utilized. Owing to the absence of phosphorus and to other excellent qualities characterizing some of these ores, they were for centuries the only ores that were mined in Sweden, and they have been the raw-material of the Swedish iron that has won world-wide renown. These ores occur especially in Sweden, but also in certain parts of Norway.

The ores of Group II. could not for a long time be utilized on account of their high percentage of phosphorus; moreover, the situation of the deposits in the extreme north of the Scandinavian peninsula discouraged mining enterprises. Some of the deposits of this group are among the greatest in the world. Thanks to the basic refining methods, they have now gained great importance, and are more and more utilized for the Swedish iron-industry; having, however, as yet chiefly given rise to an ore-export on a large scale. The ores of this group occur only in Sweden.

The ores of Group III. are chemically characterized by a high amount of titanium, which makes them very difficult to reduce. They have hitherto been made use of on a very small scale only. Vast deposits of them occur in both Sweden and Norway.

The ores of Group IV. occur only in the metamorphosed Silurian formations of Norway. They are characterized by a comparatively low percentage of iron, and have not as yet been utilized for the Scandinavian iron-industry, but preparations are going on for mining and exporting them on a large scale after subjecting them to magnetic concentration.

The iron-ores of Group V. have a limited distribution within the Silurian and Archæan rocks of the Christiania basin. They are of no practical importance.

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Group I.—The Ores of the Archæan Crystalline Schists.

Distribution.

These ores are chiefly distributed within "the ore-province of central Sweden" (see Fig. 1). Outside of that area they occur in the province of Norrbotten (Gellivare, Svappavare, etc.), in northern Sweden, and in Varanger, in northern Norway. On the southern coast of Norway, in the neighborhood of Kragerö and Arendal, there is an isolated district carrying these ores.

The Rocks.

The ore-bearing rocks of the ore-province of central Sweden are chemically and petrographically unlike the more monotonous gneiss- and granite-areas which surround them. On the whole, they constitute a quartz-feldspar formation in which purer quartzitic rocks, limestones, "*skarn*"-rocks and ore-bodies are very subordinate members. (*Skarn* is the Swedish name for rocks of varying composition, mostly consisting of lime-, magnesia-, iron- and alumina-silicates of the pyroxene-, amphibole- and garnet-groups; as secondary minerals, epidote, chlorite, biotite and talc occur. The *skarn* is scarcely an independent rock but is connected with the ore-deposits. It is formed through an interchange between the silica of the quartz-feldspar rocks and the basic constituents of the ore-formation.) Among the feldspar-rocks there are certain types, which occur constantly in different areas—viz., the amphibolitic plagioclase-rocks and the granulitic rocks composed of quartz and alkaline feldspars. Among the latter soda-granulites as well as potash-granulites are met with; also the corresponding gneisses occur. Differentiations of pure quartz-muscovite-rocks and rocks richer in alumina constitute more subordinate types.

H. Johansson, in a highly remarkable contribution to the subject,[1] has recently pointed out some characteristics in the composition of the ore-bearing formation. He shows that within certain areas a complex of granulitic and amphibolitic rocks is met with under conditions recalling the products of a

[1] *Geologiska Föreningens Förhandlingar*, Stockholm, vol. xxviii., pp. 516 to 538 (1906); vol. xxix., pp. 143 to 186 (1907). The end of this paper is not yet published.

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magmatic differentiation. The amphibolites (generally called dioritic schists) correspond to decided femic rocks with a con-

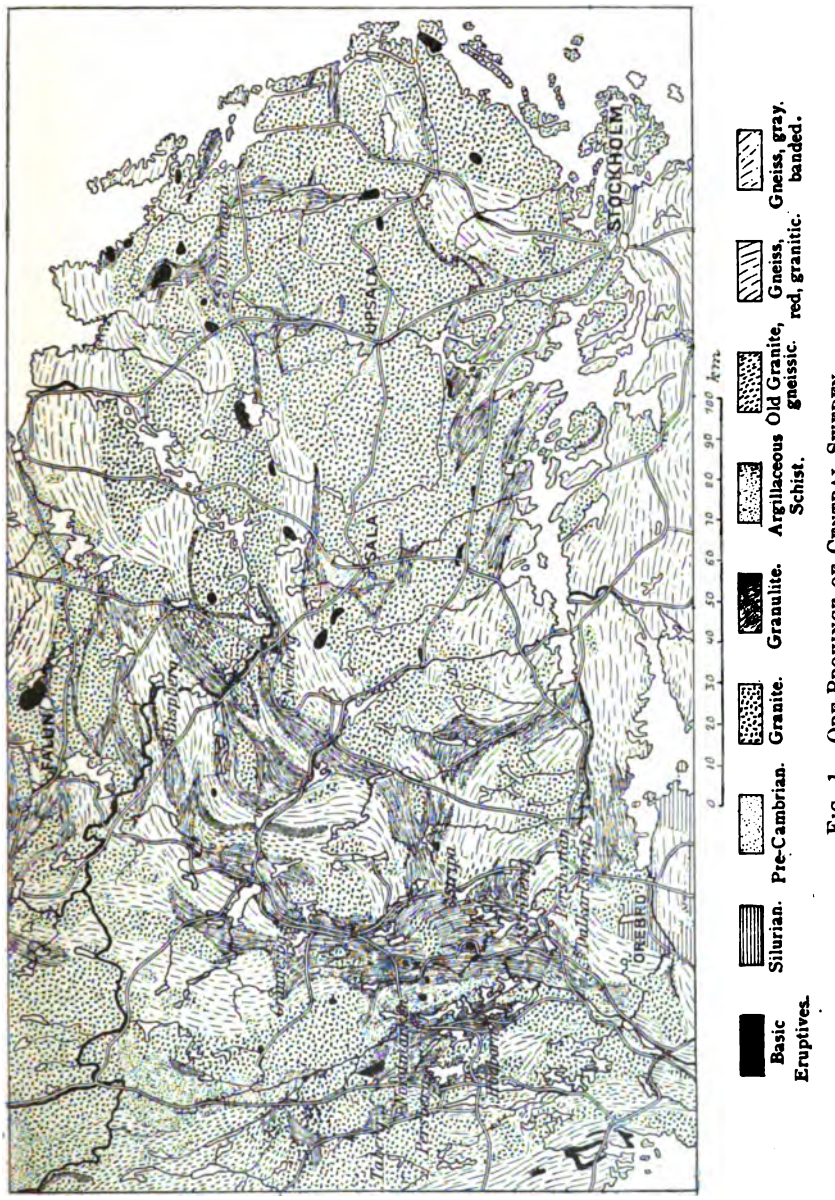

FIG. 1.—ORE-PROVINCE OF CENTRAL SWEDEN.

tent of SiO_2 not exceeding 53 per cent., while the granulites correspond to salic rock-types with at least 67 per cent. of SiO_2. The intermediate members are almost lacking.

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The amphibolites have the composition of diorites and must be considered as stretched and dynamo-metamorphosed dioritic rocks. A part of the granulites correspond chemically to known types of granite. Thus it is highly probable that igneous rocks enter, to a great extent, into the composition of the ore-bearing granulite-formation.

Chemically the granulites may be divided into two groups: one containing the types with predominating plagioclase feldspar; the other showing plagioclase- and orthoclase-feldspars in equal quantity, or with the latter predominating. On the whole, the distribution of rocks rich in soda among the ore-bearing granulites seems to be considerable.

Among the ore-bearing gneisses and *hälleflinta*-rocks also, the rocks rich in soda are frequent. The potash-rich, red "*järngneiss*," with a more granitic composition, and the "garnet-gneisses," rich in alumina, contain, on the contrary, very few ore-deposits.

The quartzitic rocks within the granulite-formation are nearly free from iron-ore-deposits, but often contain sulphide ores instead. The zones carrying these ores are often characterized through a series of quartzitic rocks containing anthophyllite, cordierite, light garnet, etc.

Structures.—According to their coarser or finer crystalline structure, the ore-bearing rocks are divided into gneisses, granulites and *hälleflintor*, the latter sometimes porphyritic. The ore-deposits prefer the fine-grained rock (granulites), but are also found in the coarse-grained gneisses as well as in the compact *hälleflintor*. As already pointed out, these rocks are very nearly related, chemically. Yet it is evident that the iron-ore-deposits are in some way connected with the rocks of granulitic structure; greater or smaller ore-deposits are found in nearly all the granulitic zones; and even the deposits in gneissic rocks lie not far from the boundary of the granulite. It is also noteworthy that the ores in the gneisses are generally not directly surrounded by true gneiss, but that the rock close to the ore-deposit assumes a more fine-grained, granulitic structure. This is observed at several ore-deposits in the gneisses of southern Dalarne (in the parishes of St. Tuna and Ludvika), as well as in Södermanland. A much more close-grained structure is also observed in certain granulites in the vicinity of the ore-bodies, where the rock may assume a compact *hälle-*

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flinta-like structure—for instance, at the mines of Sköttgrufvan, Qviddberget in Dalarne and others.

It is most probable that the granulitic structure in general is due to a recrystallization under stress and movement, within the anamorphic zone of depth. The coarser gneissic and granitic rock-material has thus been granulated, and the size of the grain reduced. This has been the case not only over the broader granulitic areas but also along the granulitic zones in the gneissic areas, which thus in a certain way correspond to shearing-zones. Simultaneously with the mechanical deformation of the rock-masses there has been also a supply of iron-bearing magmatic material by solutions, imparting to the ore-deposits their present peculiar epigenetic characters.

To the processes undergone by the granulitic formation during submersion in the anamorphic zone belongs also the intrusion of the numerous pegmatitic dikes which at so many mines (*e.g.*, the mines of Pershyttan, Grängesberg, Ludvika-district, Gellivare) penetrate the ore-deposits as well as the surrounding rocks. These pegmatites are to be considered as secretions with a low temperature of crystallization, deposited from aqueo-igneous solutions in contraction-fissures due to the cooling of the surrounding rock.

A porphyritic structure, denoting effusive surface-rocks, is observed in several *hälleflintor* (Dannemora, Utö). Other structures characteristic of surface-rocks, such as more or less evident stratification, may be seen in the rocks surrounding several ore-deposits (*e.g.*, Striberg and Utö). The stratification is locally shown as a striping in the contact between the limestone and the *hälleflinta* at Dannemora. In other ore-deposits, especially those of the *skarn*-ore type, it is common that the rock close to the ore-body exhibits an irregular or twisted structure (as at Persberg), or is traversed by numerous fissures, or even becomes brecciated (as at Gellivare). This seems to indicate that the ores were deposited in shearing-zones, or perhaps that the chemical changes connected with the replacement of the rock-constituents by the ore-substance caused a considerable reduction of volume, followed by the disturbance of the rock referred to.

A secondary form, often observable in the granulite, is the linear structure caused by stretching of the rock. This often

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shows, as at Lekomberga and Vintjern in Dalarne, a connection with the form of the ore-bodies, where the stretching is conformable to the pitch of the ore-bodies and to the linear structure of the surrounding rocks.

The true nature of the granulite-formation seems hardly to be fully explained, but it certainly contains rocks of different origin. That the amphibolites and a great part of the real granulites are igneous rocks of deep-seated origin, seems to be beyond doubt. But the residual structure of some of the *hälleflintor* makes it probable that these are igneous surface-rocks. Other surface-rocks are the limestones and the quartzites. The very rare conglomerates belong to a younger division of the Archæan than the ore-bearing granulite formation. As in older times "graywacke" was a common name for a number of imperfectly known rocks, so it is to-day with the name granulite (or "*hälleflinta-gneiss*" of the Swedish Geological Survey). Further investigations will certainly divide this group into rocks of very different nature and origin, only having in common certain structural features. The same is true as to *hälleflinta*.

In the ore-bearing district of southern Norway (Arendal, Näs, Kragerö), which in most points agrees with central Sweden, the chief rocks are fine-grained gneisses, amphibolites and quartziferous rocks.

The Rocks of the Ore-Province of Norrbotten.—The geology of this wide district is somewhat complicated. Granitic rocks, together with syenite, syenitic porphyries and crystalline schists, here dominate in it; moreover, basic eruptives of the diorite and gabbro family are frequent. A series of clastic rocks (metamorphosed sandstone and conglomerate, together with limestone) of indeterminate age is exposed in several places.

The rocks of syenitic composition—*i. e.*, real syenite, syenitic porphyry and granite—are found in Norrbotten to a larger extent than in central Sweden. The same may also be the case with the basic eruptives of the gabbro and porphyrite families. The rocks of both regions consist, however, mainly of granites, gneiss-granites and granulites—*i. e.*, quartz-feldspar rocks.

The granites are not ore-bearing, and seem not to be even indirectly connected with the formation of the ore. The granu-

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lite contains several large ore-deposits of the same type as those of central Sweden (Gellivare, Svappavare). The largest deposits are, however, genetically connected with the syenitic and porphyritic rocks (Kiirunavaara, Luossavaara, etc.), belonging to Group II. of the preceding classifications. The gabbro-rocks of the district have their special deposits (Routivare) belonging to Group III.

The rocks inclosing the iron-ores of Varanger in the north-eastern part of Norway are chiefly gabbros, amphibolites and quartz-feldspar rocks. According to G. Henriksen, who reported upon these ores, the rocks are of igneous origin, but strongly metamorphosed.

The Ores.

The ores belonging to this group may be divided into five sub-groups or types. The classification given below derives its origin from the metallurgical experiences of former times, when the Swedish iron-ores were divided, according to their behavior in the blast-furnace, into three kinds: (1) ores smelting in the blast-furnace without any flux, corresponding to the type C, in which the gangue itself contains the com-components of an easily fusible flux; (2) the quartzose ores, corresponding to types A and B, which require the addition of flux-making bases; and (3) the ores rich in lime, corresponding to type D, generally used to make suitable charges by mixture with the quartzose ores. This division, founded on the practical utilization of the ores, was adopted in 1874 by Anton Sjögren, who pointed out that the old division of the blast-furnace men corresponds to distinct geological ore-types. The apatite-ores, having at this time no great utilization, were not included in the system. In 1893, I added the apatite-ores as a special type, in which I have been followed also by H. Johansson in his discussion of the genesis of the Swedish iron-ores. Later on I divided the quartz-bearing ores into two types, A and B.

Type A. Ores containing apatite (type of Grängesberg, Gellivare).

Type B. Mixed hematite and magnetite, rich in silica and alumina and with a scaly or flaky structure (type of Lomberget).

Type C. Banded quartziferous hematites, rich in silica, and with a striped structure (type of Striberg).

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Type D. Magnetite-ores associated with silicates of lime, magnesia and alumina (type of Persberg).

Type E. Magnetite- and hematite-ores associated with limestone and dolomite (type of Dannemora and Långban).

Between the different types transitional members may be found; but these are quantitatively subordinate, and all the important deposits belong to some of the above-defined types.

From a chemical point of view, these types form a series from the most acid, containing up to 50 per cent. SiO_2, to the most basic, type E, with only a few per cent. of SiO_2.

Taking into consideration all the chemical and geological characters of the ores, H. Johansson has arranged the different ore-types in the following order: apatite-ores, quartz-ores rich in alumina, quartz-banded ores, *skarn*-ores, and ores with limestone.

Type A. Apatite-Ore.—Besides the apatite, which may be uniformly intermixed or distributed in stripes in the ore-mass, the gangue consists only of a few per cent. of silica and feldspar. These ores originally were magnetite, but in many places have been altered to specular hematite through a process which simultaneously reduced the percentage of apatite. The apatite-ores are richer in iron (of which they carry from 60 to 65 per cent.) than the other types. The ore-bodies are generally well defined against the country-rock, and are frequently bounded, at least on one side of the ore, by "*sköl*" formations consisting of biotite, amphibolite or chlorite. They belong especially to the part of the granulitic formation characterized by amphibolite and plagioclase-granulite. As in the type B also, dikes of pegmatite are frequently present to a considerable extent.

In the ore-province of central Sweden the apatite-ores are represented by a number of important deposits, containing more than half the available ore-quantity. They are confined to a comparatively narrow district in the southern part of Dalarne, containing the Grängesberg, Hammar, and Blötberg mines, and several others. In the ore-province of Norrbotten this type is represented by Gellivare.

In southern Norway this type also occurs, represented by the Nissedal deposit and by the Soldal and Lyngroth mines.

These ores are either products of magmatic differentiation in the granulite or of metasomatic replacement of the feldspar-

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rock by iron-bearing solutions in the anamorphic zone. Considering that only the first-named way of formation would explain the constant large amount of apatite in the ore, I look upon that hypothesis as the more probable.

Type B. Mixed Hematite and Magnetite.—The ores of this type, as is indicated by their territorial distribution, are nearly related to the preceding. They also show transitions to the ore-deposits of the next following type; but the regular banding is not developed, although a certain parallel structure is frequently to be seen. The ores of this type are scaly or granular in structure, and consist generally of magnetite and specular hematite mixed. The gangue is chiefly quartz, together with some mica and feldspar—the same constituents as compose the wall-rock. The percentage of iron is generally from 50 to 55 per cent., and richer concentrations are not very common.

The ore-bodies exhibit generally no defined boundaries, but show transitions into the wall-rock; frequently (*e.g.*, in the Ludvika district) the ore is disseminated as a lean impregnation in the rock.

Characteristic are the numerous pegmatitic dikes, evidently segregations of the wall-rocks, crystallized in fissures of contraction in the anamorphic zone. They correspond to the segregations of quartz, so numerous in the ore-bodies of type C.

These ores are evidently formed through injection of iron-bearing solutions in the schistose rocks and partial replacement of the rock-forming minerals; the ore thus being much younger than the inclosing rock. They are found in their typical development only in two areas—viz., in the Nora district (the Pershytte and some other mines); and in the Grängesberg district (the Lomberg, Ormberg and Blötberg mines, and the greater part of the Ludvika mines).

Type C. Quartz-Banded Ores.—This type is characterized by the regular striping due to the alternation of ore and gangue, often even as clear as that of the banded jasper in the Lake Superior ores. The ore is chiefly specular hematite; the gangue mostly quartz (*e.g.*, Striberg, V. Bispberg, Norberg in part). In several mines, however, lime-silicates in the form of garnet (Norberg, Gräsberg) or amphibole (Stripa, Utö)

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enter into the gangue; the striping is then less regular. Epidote is found as a secondary constituent.

The deposits of this type often constitute regular bed-formed ore-bodies without any *skölar*, but generally sharply defined against the wall. The iron-percentage is generally 50 to 55; but richer secondary concentrations of magnetite are found in numerous mines.

The regular striped structure has generally been considered as a primary stratification, in accordance with the view applied to the banded jasper, with which this structure fully agrees; frequently the striping is plicated and contorted. If it thus corresponds to an original stratification, these ores must have been formed through the alteration of stratified rocks with maintenance of the structure. If, on the other hand, this peculiar structure is a secondary feature, produced during the replacement-process, which seems to be more probable, the ores have been formed by a more thorough replacement of the original rock. H. Johansson, who considers the ore of this type to be, like all the other ores of central Sweden, a product of magmatic differentiation, looks on the striping as produced through "a pure physico-chemical process."[2]

The most important deposits of this type are found in several districts—viz., Striberg, Strossa, Stripa in the governmental district of Örebro, Norberg in Vestmanland, Bispberg and Grāsberget in Dalarne, Utö in the governmental district of Stockholm, etc.

Type D. The "Skarn"-Ores.—These ores are regularly accompanied by a gangue-rock consisting of lime-, magnesia- and alumina-silicates, of the pyroxene-, amphibole- and garnet-groups, together with their products of alteration, epidote, chlorite, talc, and serpentine. The gangue generally constitutes bed-like deposits, joining the different ore-bodies. The latter are generally more irregular in form than those of the preceding types, and are frequently bounded by *skölar*. They consist of magnetite containing about 55 to 60 per cent. of iron, and are especially frequent on the contacts between granulite and limestone or dolomite (Persberg, Nordmark), or between granulite and diorite (Högborn mines).

The ores of this type have often been formed by metasomatic

[2] *Geologiska Föreningens Förhandlingar*, Stockholm, vol. xxix., p. 186 (1907).

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replacement of limestone and dolomite by iron-bearing solutions, the ore and the minerals composing the gangue thus being of younger formation than the rocks.

They occur mostly in the western part of the ore-province of central Sweden, at Persberg, Nordmark, Taberg in Vermland, Dalkarlsberg and Klacka Lerberg in the governmental district of Örebro; some of the Norberg mines in Vestmanland and the Nyäng mines in Gestrikland belong also in this class, together with the ore-deposits of Arendal in southern Norway.

Type E. Limestone-Ores.—These ores are characterized by their occurrence in or together with limestone and dolomite; they frequently contain manganese. The gangue is composed of several silicates of manganese, such as knebelite, tephroite, rhodonite, and manganese-garnet. The ores are sometimes associated, as, for instance, at Långban, with manganese-ores (hausmannite, braunite).

The ore-bodies, being metasomatic deposits in limestone, are very irregular in form. They are frequently bounded by *skölar*. The source of the iron in the solutions which produced metasomatic replacement may have been either the limestone or some external rock.

These ores only occur in connection with limestone and dolomite. The most prominent representatives are Långban in Vermland, the Vikers mines, Svartvik, and the Ställberg mines in the governmental district of Örebro and Dannemora in Upland.

The Origin of the Ore-Deposits.

During the last century the ores of this group were regarded by Swedish geologists as sedimentary deposits, laid down together with the over- and underlying granulite formation. Such opinions were advocated by A. Erdmann, Anton Sjögren, A. E. Törnebohm, B. Santesson and others; among the Norwegian geologists Vogt has with eagerness developed this theory.[3]

[3] See, for instance, J. H. L. Vogt, De lagformigt optraedende jernmalmforekomster, *Geologiska Föreningens Förhandlingar*, vol. xvi., p 275 (1894); Dunderlandsdalens jernmalmfelt, *Norges Geologiske Undersögelse*, No. 15, pp. 56 to 63 (1894); Om de lagrade jernmalmsfyn-digheternas bildningssätt, *Wermländska Bergsmannaföreningens Annaler* (1896). The same opinion has also been expressed by several foreign geologists, for instance, De Launay, *Annales des Mines*, Tenth Series, vol. iv., pp. 49 to 209 (1903), as well as in the German treatises on Ore-Deposits by Beck (1901), and Stelzner-Bergeat (1906).

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Only with respect to the Gellivare ores, opinions were much divided, and several geologists—*e. g.*, Lundbohm, v. Post and Löfstrand—believed them to be of igneous origin.

Since 1890, the present writer has in several papers argued that metasomatic processes undoubtedly played a prominent part in the formation of these ores, and has been able to point out several analogies with the iron-ores of the Lake Superior region.

In this paper I shall attempt to show that the metasomatic processes must have taken place, not in the surface-zone but in the anamorphic zone, and that the ores bear fully, in their mineralogical features and association, characters of formations of the deep-seated zone.

In several cases, it may not be possible to determine whether the original iron-bearing material was the product of primary magmatic differentiation, as in the apatite-ores, or iron-bearing magmatic solutions producing metasomatic deposits, as in the lime- and *skarn*-ores, or possibly altered chemical sediments, as in the quartz-banded-ore type.

The iron-bearing solutions may frequently have been of magmatic origin, thus carrying iron-bearing material from below; or it may be that the very small amount of water contained in the rocks was sufficient, under the condition of dynamic metamorphism in the anamorphic zone, to collect and concentrate the iron particles. The occurrence of ore-deposits in connection with surface-rocks, above pointed out, and their absence in the greater granite laccolites seems to prove that the deposits are formed in a depth less than that in which the granite consolidated, but still in the anamorphic zone.

Arguments Against the Sedimentation Theory.—That these deposits are not sedimentary is indicated by the fact that the surrounding rocks are igneous. So long as the granulite of Grängesberg and Gellivare was considered a sedimentary rock, it was possible to ascribe the same nature to the ore-deposits. But this foundation of the sedimentary theory seems more and more to give way.

Again, the form of these ore-deposits differs as widely as possible from that of stratified bodies. They have been termed lenses, stocks, lineals, etc. In general, they are much more irregular than is consistent with a sedimentary formation.

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Sometimes these ores divide or branch into the surrounding rock (see Fig. 2, showing the central part of Dannemora), a feature which does not agree with any known form of primary sedimentation, but must be interpreted as secondary. At other places, the ore incloses portions of the surrounding rock, which sometimes take the form of irregular sinuous bands, cutting obliquely through the ore from the hanging-wall to the foot-wall. (See Fig. 3, showing the central part of Grängesberg.) In such cases it appears that the overlying and the underlying rock, together with the narrow partition-walls

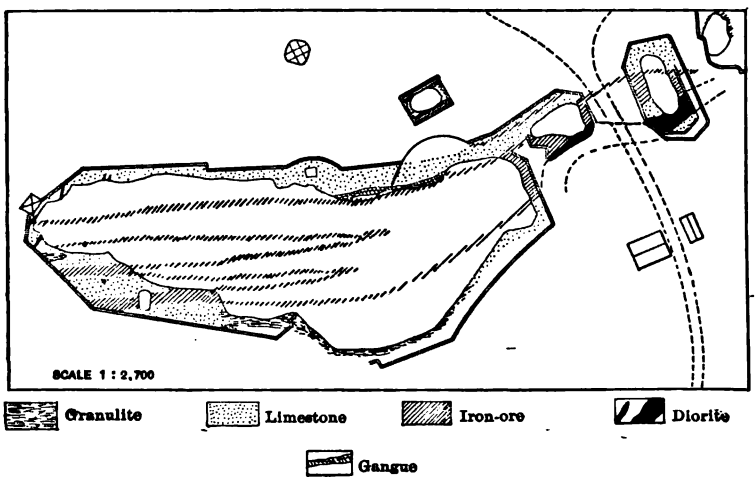

FIG. 2.—DANNEMORA MINE, CENTRAL PART.

between the ore-lenses, form "a continuous whole, pre-existing to the ore." This mode of occurrence also is incompatible with sedimentary deposition.

In a few cases only, a structure resembling primitive stratification is met with; *e. g.*, in the banded quartziferous ores (type C), which for this reason possess special interest. The ores rich in silica and alumina (type B) sometimes present a schistose structure; but this is without any doubt a foliation caused by pressure.

Even where the stratified structure is present, we are not justified in concluding that the material is primitive; for in this case also, though retaining the original structure, it may have been subjected to subsequent metasomatic transformation.

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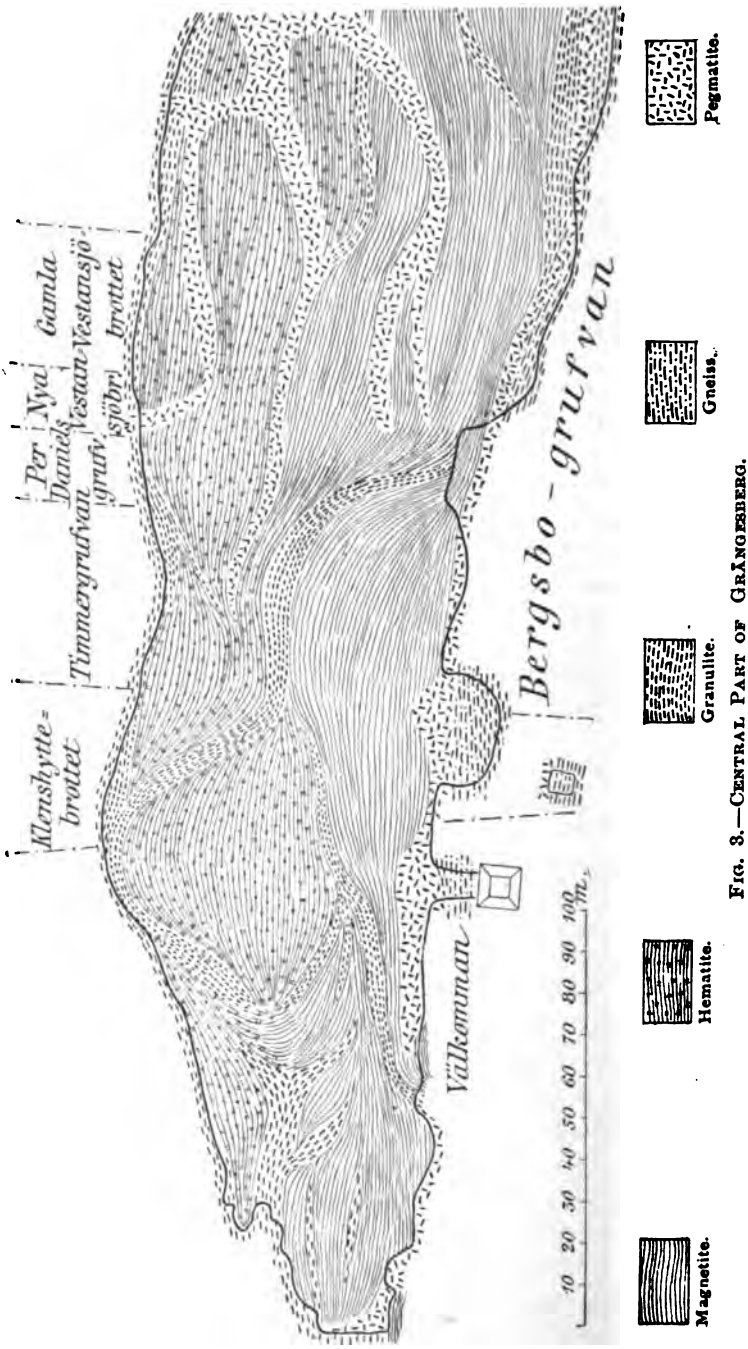

FIG. 3.—CENTRAL PART OF GRÄNGESBERG.

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Thus the banded quartziferous ores may have originally consisted of alternating bands of a carbonate and of amorphous silica, since altered or replaced *in situ.*

Concentration and Transformation Within the Deep-Seated Anamorphic Zone.—To this zone the rocks were transferred during the period of the plication, by which evidently a part were folded to a considerable depth, being at the same time subjected to dynamo-metamorphic alterations, which in many cases determined their present characters.

During this period the ores and the gangue were formed by thermal iron-bearing solutions acting under high pressure. To what degree these solutions were magmatic, carrying ore-substance from below, or to what degree the small amount of water contained in the rock was active, it is not possible to determine. In either case, the process was different from the action of solutions circulating in open channels. It consisted in a solution of the rock-substance, which was intensified by the stress and friction, according to the principle of Riecke, and also in an accumulation and concentration of the ore, the surface-tension operating to unite particles of the same substance.

The ore-material, participating in the plication process, with its upheavals, folding and dislocations of the strata, has suffered some mechanical changes. One of the effects which, in many cases at least, may be ascribed to the mechanical folding is the peculiar overlapping which many ore-bodies present. This may be a primitive form of metasomatic deposition, but may also be considered as a result of the mechanical displacement of an ore-layer. Through a number of inclined or even vertical planes of dislocation, the deposit has been cut into pieces, which have then been somewhat displaced in relation to one another. Often these dislocations can be pointed out only with difficulty, or not at all; they are more obvious when the iron-ore occurs associated with a limestone-bed. (See Fig. 4, showing the plan of Ställberget.) The whole limestone horizon, with its accompanying iron-ore, is here fractured into several lenses, oblique to one another. The planes of dislocation have been effaced by the recrystallization of the granulite, and the schistose structure thereby developed, with its lamination running obliquely to the different lenses, which gives the

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appearance of being situated on different levels of the stratigraphical series.

Another mechanical effect of the rock-plication is the stretching of the surrounding rock, by which it has assumed a linear structure, a system of smaller folds, with the axis of folding parallel to the stretching, having often been formed at the same time. In many cases (*e. g.*, the Lekomberga mine in the parish of Ludvika, and the Smedje and Mossaberg mines in Striberg), a connection between the stretching of the rocks and the pitch of the ore-bodies is observable, the axis of the stretching coinciding with the direction of the pitch. Yet it cannot be assumed as beyond dispute that this connection is in every case

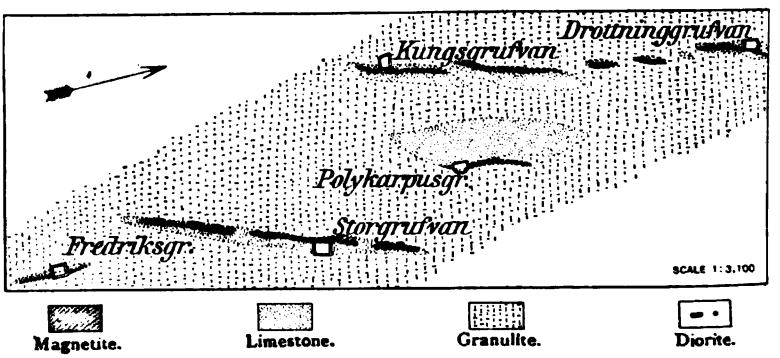

FIG. 4.—STÄLLBERG MINES.

due to a stretching of the ores themselves; it may also be explained as produced by the ferriferous solutions, chiefly following the directions indicated by the folds, formed simultaneously with the stretching of the rock.

The stretching of the rocks has given rise to the characteristic form presented by the Swedish ores of this type, which form is evidently due to a factor acting in a vertical direction. This form is represented in Fig. 5, which is a longitudinal section of part of the Svartvik mines, according to B. Santesson. Sometimes this form will be developed into such an extreme type as that of the mine of Stora Malmsjöberg, in which the ore mined in 1898 had, according to H. Sundholm, a length of 15 m., a breadth of 12 m., and a depth of 150 meters.

The Chemical Changes.—These have been much increased, not

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only by the high temperature in the anamorphic zone, but also by the stress and mechanical deformation, which tends to increase the solubility of the ore-material, as well as of the rocks.

The more easily soluble limestones were especially adapted to take up the ore-deposition; and generally great changes and transfers of the ores to secondary places of deposition have taken place in them, depending on water-courses, impermeable sub-strata, etc. The concentration has occurred especially along folds, fracture-zones, fissure-systems and contacts. Thus the ores assume the irregular, secondary forms characteristic of

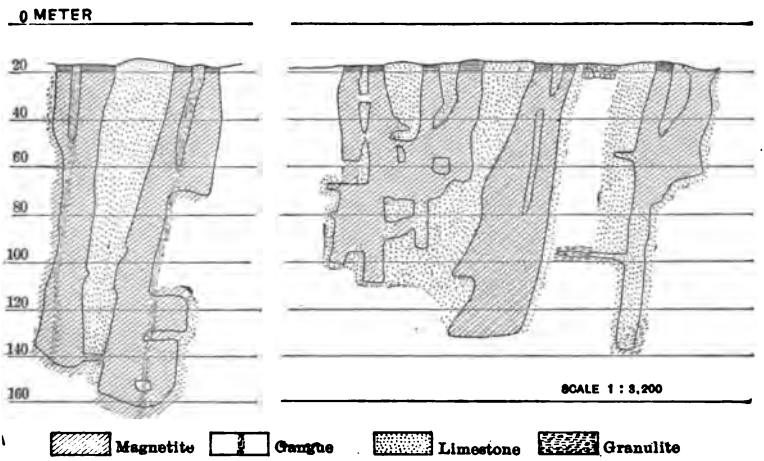

FIG. 5.—LONGITUDINAL SECTION OF THE SVARTVIK MINES.

metasomatic deposits, as shown in Fig. 2, representing the central part of Dannemora.

Other instances of concentration by solutions are offered by the fairly numerous cases in which the ore proves to be younger than dikes which traverse it. Such an instance is the Timansberg, a deposit of type D, which is traversed by minor dioritic dikes. Of these, C. H. Vrang writes that "in their vicinity the ore increases considerably in thickness."[4] In the Krangrufva, in the Persberg district, the rich and pure ore is chiefly found on one side of a large diorite dike; a dislocation is out of the question, for the ore-bearing layer is also found on the other side of the diorite, but without equally distinct ore-con-

[4] *Geologiska Föreningens Förhandlingar*, Stockholm, vol. ix., p. 244 (1887).

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centration. H. V. Tiberg has described how, in the Gustavus mine of the Långban district, a deposit consisting of specular iron-ore and magnetite, with gangue, continues from the surface to a flat system of diabase apophyses occurring at a depth of from 54 to 60 m., on which the ore spreads like a cake, while below the diabase-dikes the rock is dolomite only. In such cases the intrusive dikes have evidently preserved the underlying rock from transformation. A similar instance from Dannemora is mentioned by A. E. Fahlcrantz,[5] who says that over a dike of *hälleflinta* (felsite-porphyry) a band of iron-ore a few inches in thickness was met with, accompanying the *hälleflinta*. In this case, however, it is probable that the ore is a more recent formation; for the majority of the Dannemora ores are certainly older than the felsite-porphyries.

Many limestone-beds have been largely, or even wholly, transformed into ores, especially of type D, the gangues of which, consisting of calcium- and magnesium-silicates, clearly indicate their origin.

At Utö it is questionable whether the concentration of the most prominent ore-deposit is not connected with the two traversing pegmatite dikes. The largest and deepest mines, which have followed the deposit down to a vertical depth of more than 200 m., are situated between these two pegmatite dikes; moreover, on the outer sides of the pegmatites there is a continuous ore-mass, which has been followed down to a comparatively great depth. At some distance from these dikes the ore has everywhere been less thick and less concentrated, and has, therefore, been mined on a small scale only. While the pegmatite dikes evidently originated at a great depth, it follows that the ore-concentration could not have been accomplished earlier than the submersion in the deep-seated zone.

The numerous mines of the Grängesberg, Blötberg, Fredmundsberg and Gräsberg districts offer good opportunities to observe the relation between the pegmatites and the ores. The former occur here as dikes, partly traversing the ores, and partly parallel to the stratification, but in a manner which indicates that they are of later formation than the ores. Coarsely crystalline magnetite is often found in the pegmatite veins, in-

[5] *Kunglig Vetenskapsakademiens Handlingar Bihang*, Stockholm, 4 (1876).

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dicating that the aqueo-igneous solutions giving rise to the pegmatite originated from the same source as the iron-ore.

In this connection attention may be called to the not uncommon fact that the ores of this type occur along contacts, generally between limestone and granulite, but also along the contact

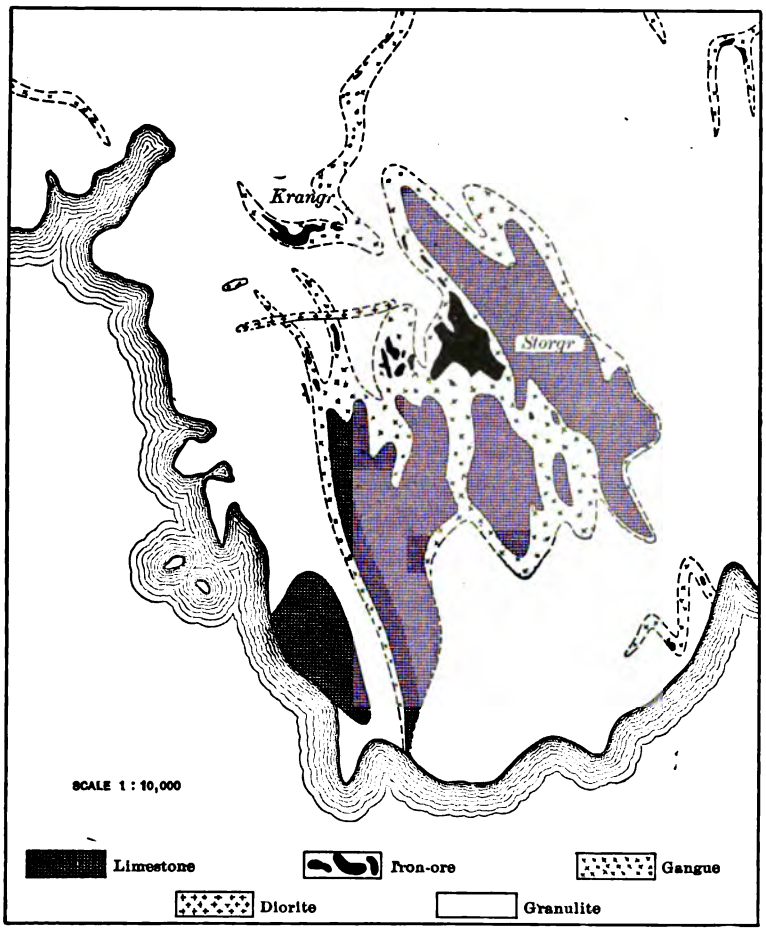

FIG. 6.—PERSBERG.

with intrusive rocks. A marked instance of the former mode of occurrence is seen at Persberg (Fig. 6), where the upper ore-bearing horizon occupies a basin with granulite in the foot-wall and dolomized limestone in the hanging-wall. Similar instances are found in several other ore-deposits of type D; *e.g.*, the Gåsgrufve mine and the Nordmark mines in Vermland, and Sten-

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ring and Ramhäll in Upland. In such cases the situation of the ores along the contact can hardly be explained by assuming them to be of primary origin; the only explanation possible is that the ores have been precipitated along the contact from ferriferous solutions.

The same explanation presents itself in such a case as that of the Högborn district in Örebro, where the most important deposits occur on a certain level between a diorite mass and the granulite (Fig. 7). If the ores were assumed to have been laid down as a sedimentary deposition on a certain level within the

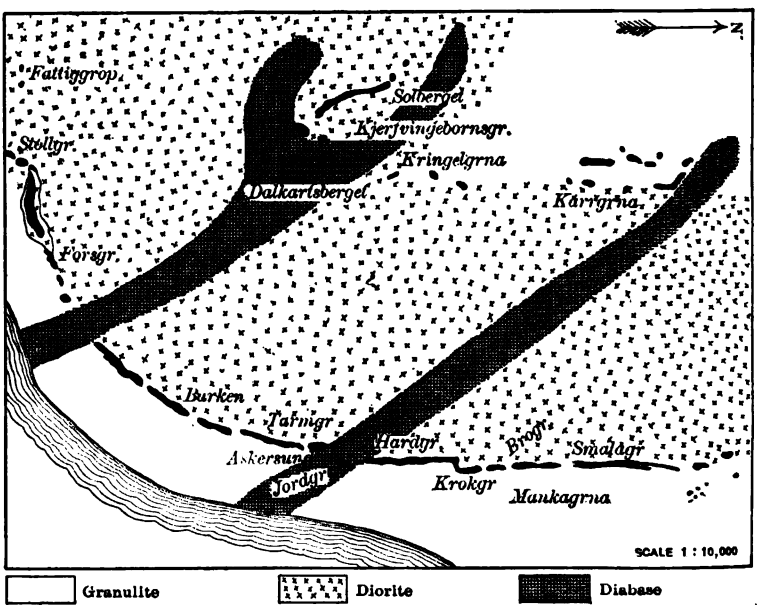

FIG. 7.—HÖGBORN MINES.

granulite, it would be necessary to explain the fact that the diorite had been injected on this very level and formed a laccolite there. The assumption that the ores are younger than the dioritic laccolite, and that they have been precipitated from ferriferous solutions, removes this difficulty. The most satisfactory view, therefore, is that the ores along such contacts are epigenetic formations.

To this period also belongs the formation of the gangues, which are the result of the silicification that takes place in the deep-seated zone. Whole layers of limestone have, through

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the interchange of constituents, been transformed into lime-magnesia-iron-silicates (*skarn*-deposits). If the limestone has already undergone a dolomitizing process, or if Mg is added by the solution, the chief alteration-products will be amphiboles, which are rich in magnesia; if the limestones are comparatively pure, pyroxenes are formed. Though the garnet of the gangues may often be an alteration-product of pyroxene or amphibole minerals, it may, however, frequently be of primary formation, depending on the chemical composition of the solutions and the transformed material.

Transformations in the Surface-Zone.—When the erosion and removal of the overlying Cambro-Silurian strata exposed the Archæan rocks, the ore-deposits were subjected to the influence of catamorphic agencies.

Among the transformations of this period we have to note the formation of many *skölar* of chlorite and talc (soapstone) by the decomposition of the pyroxene and amphibole of the gangues or the granulite of the wall-rock. Many *skölar*, too, owe their origin to the decomposition of intrusive greenstone-veins.

Indeed, the whole mass of the gangues may, under certain circumstances, be changed, quite new ore-types being the result of the transformation. A common phase of this transformation is the occurrence of epidote, quartz and calcite in the gangue; if the alteration proceeds to a certain degree, hydrated minerals of the talc and chlorite groups are formed. As has been shown by H. V. Tiberg,[6] the gangue in the Taberg mines in Vermland, the ore of which is, in its upper part, markedly talcose, has been formed by metasomatic transformation of augite and amphibole. This transformation reaches only as far down as 320 m., at which depth the ore is cut nearly horizontally by a fissure which yields plenty of water. The same author adduces, also, the Alabama mine in the Persberg district as an instance of similar transformation; in its southern part the ore is talcose, but in the northern part the original gangues, pyroxene and amphibole, are still found. No doubt the ores of Dalkarlsberget, as well as all the ores belonging to the so-called "Rösberg type" of B. Santesson, ought to be in-

[6] *Wermländska Bergsmannaföreningens Annaler* (1903).

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terpreted in the same way, as being derived from type D through alteration.

Several other transformations may be viewed as comparatively recent and, consequently, belonging to this period. The ores of the Striberg type are often found to lose their characteristic bandedness in the direction of the strike, the quartz being replaced by a somewhat porous magnetite, sometimes also by calcite. Concentrations of richer magnetite often occur among the ores of this type; they are generally accompanied by quartzose segregations or chloritic *skölar*.

Simultaneously with this, a concentration of the iron may take place, SiO_2 being dissolved by means of alkaline carbonates and the silica replaced by ferric oxides. It is by such concentration that, for instance, the rich ore-deposits have been formed which are often met with in folds; *e. g.*, in Nordmark and in the Högborn mines in Vermland. In localities where very thick deposits of ore not enriched, and retaining the primitive structure, occur in folds, as at the Stripa mine, the abnormal thickness must be ascribed to mechanical deformation.

Among the Gräsberg mines, which are worked on a folded layer in the form of a trough sloping NNW., there occurs in the Bolag mine a highly concentrated magnetite in the fold, the rest of the ore being a poor hematite with quartz bands.[7] That a concentration process has taken place here is beyond doubt, though it is not possible to determine to what period it should be assigned.

On a still larger scale, a similar transformation has taken place in the Bispberg mine, producing a rich, pure magnetite, mined in the deeper levels of this mine, while in the upper parts the ore was a typical low-grade quartz-banded hematite.

Such transformation on a large scale of specular hematite into magnetite has long been known from several ore-deposits; *e. g.*, Norberg, Striberg, Gellivare. In the ends of the ore-bodies and in the sides contiguous to the surrounding rocks the transformation is most advanced, but also the interior portions of the ores consist of a mixture of specular hematite and magnetite. The cause of this transformation is not fully understood; but it is evidently a reaction, depending on mass-

[7] H. Sundholm, *Jern-Kontorets Annaler*, vol. liii., p. 162 (1898).

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action, and continuing to a certain limit, where equilibrium ensues. It looks as if the reaction proceeded from the surrounding or traversing silicate-rocks; and it has been conjectured that the alteration has been produced by organic substances (humic acids) contained in the surface-waters descending along the walls of these rocks. Even though it might be supposed that these organic substances had the power of reducing specular hematite to magnetite, this explanation is not very satisfactory; for the transformation that has taken place is not a reduction of hematite to magnetite with retention of the structure of the former, but a solution and recrystallization of the iron-ore. One might rather suggest the action of alkaline solutions proceeding from the silicate-rocks, or some other reagent.

The change from hematite to magnetite is reversible; and in some places we meet with transformations on a large scale of magnetite into hematite. Of the anhydrous iron oxides, magnetite is more stable in the deep-seated zone, hematite in the surface-zone; and it seems safe to assume that the last-mentioned alteration belongs to the surface-zone. Such a transformation is found in the Grängesberg mines, where the ore close to the foot-wall consists of a scaly hematite low in phosphorus.

The Grängesberg and Norrbotten Deposits.

A separate position should be assigned to those ores of the type which are chiefly represented by the large deposits of Grängesberg, in central Sweden, and Gellivare, in Norrbotten. They diverge in some points from the majority of the ores of the Archæan schists; and their characteristic properties seem to be most satisfactorily explained by assuming them to be transformed basic segregations in gneiss-granites (orthogneisses).

In sketching the geology of Grängesberg I follow chiefly the notes given by H. Johansson.[8] In this district two ore-types are represented. The ores of the Lomberg-Risberg, and Ormberg mines, occurring in a real kali-feldspar-granulite, here represented by a reddish rock of uniform grain, have all the

[8] *Geologiska Föreningens Förhandlingar*, Stockholm, vol. xxvi., pp. 361 to 363 (1904) and vol. xxix., pp. 174 to 176 (1907).

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characters of the type B. To another type belong the vast ore-stocks of the Export mines, surrounded by a gray plagioclase-granulite, often, however, containing amphibole and inclosing numerous amphibolitic segregations. To the same class as the ores of the Export mines belong those of the Hammar mine, where the amphibolitic development of the rock is still more obvious, and several minor deposits; *e. g.*, the Långblåfall mines, which occur associated with basic rocks of dioritic type. It seems that these rocks are genetically connected with the gneiss-granite formation, developed immediately to the east of the Export mines. This formation varies in structure; in the central portions it is purely granitic and coarse-grained; nearer to where it borders on the granulite it becomes flaky, and the grain grows finer, only scattered porphyritic grains of feldspar remaining, till it passes into the fine-grained rock which incloses the ores; this rock becomes in places amphibolitic and dioritic (the mines of the Långblåfall mine.) On the whole, the ores, as well as the rock-complex inclosing them, present a peculiar lack of homogeneity in their composition, indicating segregations and concentrations in a magma. To the same class I refer also the Lekomberga deposits, likewise situated in a gneiss-granite (orthogneiss) formation.

The ore-deposits in the Export mines consist of composite stocks, the different parts of which are separated by partition-walls consisting of the same materials as the adjacent rock. (Compare Fig. 3.) The ores exhibit sharply-defined boundaries, and, in the composition of the gangues, SiO_2 and Al_2O_3 are subordinate to CaO and P_2O_5 (belonging to the apatite); the percentage of P varying from 0.8 to 2 per cent. and being sometimes considerably higher. Generally, it is greater at the hanging-wall and smaller at the foot-wall. Titanite occurs abundantly in some of the ores.

In a still higher degree the gneiss-granite (orthogneissic) character of the rocks in Gellivare is manifested. This is pointed out in a convincing manner by Hj. Lundbohm, in his notes of the geological conditions of this deposit,[9] though

[9] *Sveriges Geologiska Undersökning*, Ser. C, No. 111 (1890), No. 127 (1892), and elsewhere.

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he speaks with great caution on the question of the nature of these rocks. A true conception of this kind of crystalline schists was not at the time of his publications so well established as now.

The red gneiss dominant in the mountain of Gellivare is a rock with mostly allotriomorphic structure; where a parallel structure is found it is evidently a phenomenon of stretching. A porphyritic structure is found in several places. That the feldspar is plagioclastic indicates a rock rich in soda; the percentage of titanite is also remarkable.

Amphibolites, sometimes of dioritic type, occur in great numbers in the form of flattened lenses. Dikes of granite rich in soda occur, sometimes parallel with the stratification of the gneiss and with the strike of the ores, partly as masses, and differ from the gneiss chiefly or exclusively through their coarser structure. In some parts of the mountain rock-members resembling these granites rather than the gneiss inclose ores. The whole is a complex of rocks of common magmatic origin with a heterogeneity of composition depending on an advanced differentiation.

The Gellivare ores are in part surrounded by gangue-rocks, rich in ferromagnesian silicates, iron oxides, apatite, and sometimes titanite, which give evidence of the interchanges of material that have taken place between the ores and the surrounding rocks. These gangue-rocks are also in many places characterized by a brecciated structure.

The structure of the ore is coarsely crystalline, allotriomorphic. Among the structural peculiarities are the stretching phenomena, in the ores as well as in the rocks, which often give rise to a cleavage.

The latest secondary alterations concern the degree of oxidation of the iron. If the ores have been segregated from a magma, they have certainly been originally magnetites. However, Gellivare as well as the Export mines in Grängesberg consist to a large extent of specular hematite, which, consequently, is a secondary formation within the surface-zone. The formation of the specular iron-ore seems in places (especially in the foot-wall of the Bergsbo mine in Grängesberg) to have been attended with a far-advanced process of dephosphorization.

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There are so many points of agreement between Gellivare and Grängesberg that it cannot be doubted that they have been formed in the same way. The heterogeneous nature of the surrounding rocks, the high percentage of apatite in the ores, often exceeding that of silicates, and the size and peculiar form of the ore-bodies, form the chief points of agreement.

The view expressed here concerning the nature of the ores of Gellivare and Grängesberg is in close accordance with the view advanced before by Löfstrand.[10] Similar opinions, especially concerning Gellivare, have been more or less positively advanced by several other authors in the lively discussion on the formation of these deposits which was carried on at the beginning of the nineties, in the last century, though at that date the problems concerning the Archæan rocks had not yet been proposed in a form that made a clear formulation of these views possible.

Also, the analogies between the deposits of Gellivare and those of Kiiruna have been repeatedly pointed out by Lundbohm, Törnebohm, Holmquist, and others, with the remark that Gellivare is to be regarded as a transformed Kiiruna.

Analogous Deposits.

The ores of this group are well represented within the Archæan series of the United States and Canada; indeed, all the different types mentioned above are found there. This feature, that the same ore-types may be recognized in areas so far apart as North America and the Scandinavian peninsula, strongly indicates that these types correspond to certain genetical conditions. Of course, it may happen that other types, not represented in Scandinavia, may occur in the United States and in other areas. Only a few instances of the parallelism between the Scandinavian ores of this group and the Archæan ores of the United States and Canada may here be mentioned.

The ores of *Type A*—*i. e.*, apatite-ores—are typically represented by the ores of the Mineville group, Lake Champlain district. I visited these mines in 1891 and was struck by the resemblance of the ore to the ores of Gellivare and Grängesberg.

[10] *Geologiska Föreningens Förhandlingar*, Stockholm, vol. xvi., pp. 136, 137, 147, etc. (1894).

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In hand-specimens the ore is hardly to be distinguished from the ore of the Swedish mines. Not only the mineralogical composition of the ore but also the form of the ore-bodies, the included "horses" of the wall-rock, the dikes of pegmatite and the amphibolitic segregations of the country-rock, offer strong analogies.

The ores of *Type B*, rich in silica and alumina, correspond in part to the Archæan ore-deposits of New York and New Jersey, as described in these *Transactions* by Frank L. Nason[11] and in several reports of the New Jersey Geological Survey. In the late description by Arthur C. Spencer,[12] who points out the connection of the New Jersey ores with the pegmatite-dikes, the agreement with the Swedish ores of this type is still plainer.

The quartz-banded, siliceous ores of *Type C* recall at many points the ores of the Lake Superior region, especially those of Archæan age on the Vermilion range. But also the ores of the older Huronian series of the Marquette and Crystal Falls districts agree in many points with the Swedish examples of this type. In comparing the Swedish and the American ore-deposits it must be noted that the unconcentrated ore with from 45 to 50 per cent. of iron corresponds to the banded jasper of the Lake Superior district, with a considerably lower percentage of iron, while the concentrations of magnetite in the Swedish mines correspond to the ores actually mined in the Lake district.[13]

Type D, the *skarn*-ores, have also their equivalents among the Archæan iron-ore deposits of New York, New Jersey and Pennsylvania. To the same type belong also the deposits of the Cranberry district, North Carolina-Tennessee, described by Arthur Keith.[14] The description so closely corresponds to the Swedish ores of this type, that it may be verbally applied to some of them. As already observed, I fully agree with Keith

[11] *Trans.*, xxiv., 505 to 521 (1894).

[12] *Mining Magazine*, vol. x., pp. 376 to 381 (1904).

[13] In 1891 I had the opportunity to visit the Marquette and Menominee districts; and after my return to Sweden, I pointed out in an address read Nov. 5 before the Geological Society of Stockholm, the analogies between the ore-deposits of Lake Superior and several of the Swedish iron-ores. The same was further stated in a paper on the same subject in 1893. *Geologiska Föreningens Förhandlingar*, Stockholm, vol. xiii., p. 578 (1891), vol. xv., p. 473 (1893).

[14] *U. S. Geological Survey, Bulletin* No. 213, pp. 243 to 246 (1902).

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concerning the origin of these ores as depositions through replacement by iron-bearing waters in the deep zone. Of the same type are also some of the deposits of eastern Ontario, in Canada. The agreement may be so close as to include mineralogical details also. In 1891, when visiting the Tilly Foster mine, Putnam county, N. Y., I was surprised by the mineralogical association in this mine, quite corresponding to Nordmarken in Wermland. The ore is in both places crystalline magnetite, intermixed with a gangue chiefly consisting of amphibole and pyroxene; magnesian silicates of primary and secondary formation are frequent. In both places occur as characteristic minerals fine crystallized magnetite, calcite, yellow titanite and transparent apatite; moreover, the alterations are the same. The minerals of the chondrodite group, so finely crystallized in the Tilly Foster, were at that time not known in the Nordmark mine, but were found there later, thus making the agreement still more complete.

Ores belonging to *Type E*—*i. e.*, associated with limestones—seem to be comparatively scarce in the United States. Still, such ore-deposits are reported from Franklin Furnace, N. J., and from some other mines; they occur also in Ontario.

Group II.—The Ores of the Porphyries (Keratophyres).

The province of Norrbotten contains many times as much iron as all the rest of Sweden. Some of the deposits in this province may be reckoned among the largest in the world. The export from Kiirunavaara commenced as late as 1902, when the railway to Narvik on the fjord of Ofoten was opened; in the following year the exploitation of the less considerable neighboring deposit of Tuollavaara also began.

Geologically, the iron-ore deposits of Norrbotten are of three kinds: (1) the ores of the crystalline schists, which embrace the deposits of Gellivare and Svappavare, treated in the preceding section; (2) the ores of the Kiiruna type, connected with syenitic porphyries; and (3) ores connected with basic igneous rocks. These ores will be treated in the next section. (See Fig. 8.)

Topographically, the more important deposits may be divided into four groups: (1) Gellivare, embracing Malmberget and Koskulls Kulle and a few copper ore-deposits north of the Lina-

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elf; (2) Svappavare, Leveäniemi and Mertainen, situated between the Kalix and the Torneå rivers; (3) a group in the vicinity of Lake Luossajärvi, embracing Kiirunavaara, Luossavaara and Tuollavaara; (4) Ekströmsberg, which belongs to the basin of the Kalix river. Besides these, minor deposits, for the most part only imperfectly known, are scattered all over the wide area.

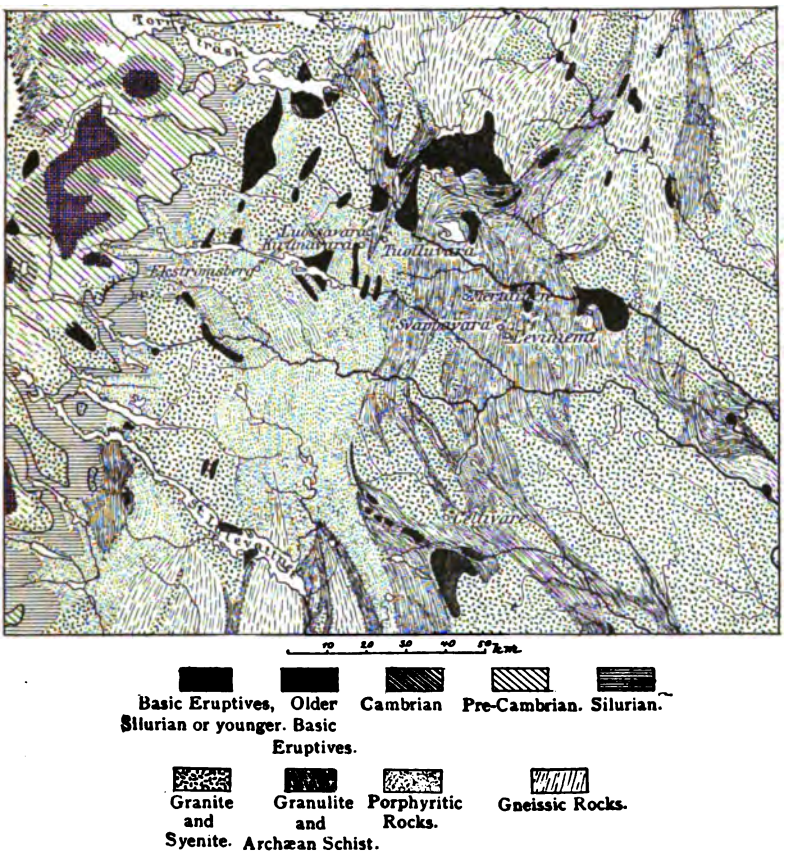

FIG. 8.—ORE-PROVINCE OF NORRBOTTEN.

Kiirunavaara, Luossavaara and Tuollavaara.

The iron-ore deposit of Kiirunavaara is undoubtedly the largest deposit of ore found in Europe. The neighboring deposits of Loussavaara and Tuollavaara are geologically of the same nature, though smaller. The first two of these deposits have been known for more than two centuries. Luossavaara is

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mentioned as early as about the year 1690, Kiiruna not before 1736; a description of both is given in the Report of the Government Mining Inspector for 1751. The first survey was made shortly before 1760. Tuollavaara, being concealed under a thick moraine, was not discovered until 1897. (See Fig. 9.)

The ores in question prove to be genetically connected with a group of eruptive rocks of syenitic composition, and characterized by their high percentage of soda. These rocks show the structures of deep-seated as well as of vein-rocks. They are intruded in a sedimentary, partly clastic, complex of strata, including conglomerates and semi-crystalline schists.

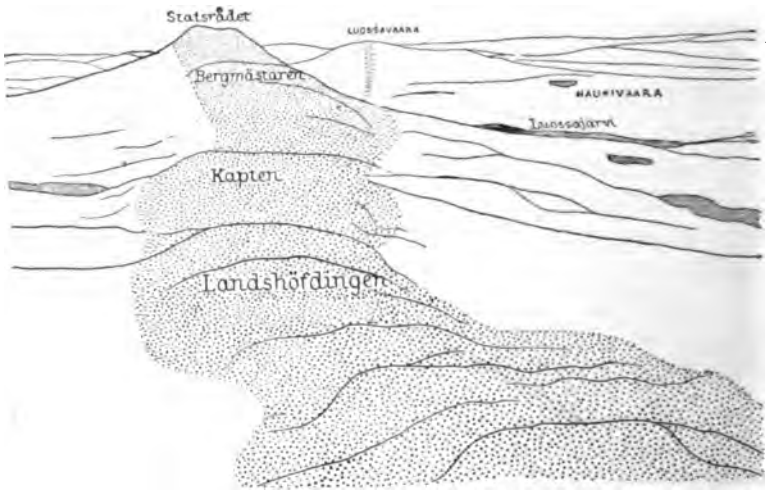

FIG. 9.—PART OF ORE-DEPOSIT OF KIIRUNAVAARA (LOOKING NORTH).

The porphyritic rocks were, for a long time, regarded by the Swedish geologists, Hummel, Gumaelius, Fredholm, and others (in analogy with the case of central Sweden), as a sedimentary *hälleflinta*, and the stratified rock-complex in which they occur was called *hälleflinta*-schist. In 1889, however, Törnebohm pointed out the porphyritic nature of the so-called *hälleflinta*, and afterwards the *hälleflinta*-schists were found to consist of partly clastic rocks.

The ores of the three deposits form stratiform masses of considerably greater length than breadth—the length of the Kiirunavaara deposits is about 2.8 kilometers.

The ores are immediately surrounded by intrusive rocks of

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porphyritic development which, on account of their composition, are to be referred to the soda-syenite-porphyries. They have also been called keratophyres (H. Bäckström); however, as this name is applied to effusive rocks, and the effusive nature of the Kiiruna porphyries seems to me at least questionable, I prefer, for the present, the name porphyries. Two kinds of porphyry may be distinguished: one more basic, occurring chiefly in the foot-wall, but partly also in the hanging-wall of the ore, and one more acid, often developed as quartz-porphyry and occurring in the hanging-wall of the ores of Kiirunavaara and Luossavaara, around Tuollavaara, etc. The basic porphyry is closely connected with the syenitic rock which accompanies it.

The Syenite.—This is a soda-syenite, the chief mass of which is a soda-feldspar. Secondary basic minerals are present in abundance. The structure is eugranitic. This soda-syenite shows gradations into the porphyry of the foot-wall, with which it is closely allied in composition.

The Porphyry of the Foot-Wall.—This rock presents, microscopically as well as macroscopically, a fluidic structure with a trachytoidal arrangement of the feldspar of the ground-mass; sometimes, also, spherulitic structures are observable. The primary structures are, however, frequently difficult to distinguish, being in part totally obliterated by the alteration of the rock. The basic constituents are almost wholly altered into amphibole, epidote, and chlorite. Magnetite seems to occur in two generations: one primary, the other of later immigration. As fissure-minerals, indicating a secondary action of pneumatolytic nature, occur amphibole, titanite, apatite, and magnetite. In the contact-zone the fissures sometimes form cavities a decimeter in diameter, filled with the said mineral combination.

The Porphyry of the Hanging-Wall.—This is considerably more acid (containing 10 per cent. more of SiO_2) than the syenite and the porphyry of the foot-wall, which circumstance places it among the quartz-porphyries. Quartz occurs in the ground-mass partly as so-called "*quartz globulaire*," but in larger quantities where the ground-mass has undergone a recrystallization. Here, too, a secondary generation of magnetite can be observed. The primary basic mineral constituents are completely altered,

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having produced amphibole, epidote, and chlorite. The rock shows, even macroscopically, a distinct fluidic structure.

Segregations of pure magnetite, mostly in rounded pieces, are worthy of notice; these segregations sometimes show a concentric structure and at times inclose grains of the feldspar of the porphyry. When the fragmentary character is more distinct, they are probably portions of segregations, solidified in depth at an earlier date, which have been partly rounded by resorption. These segregations have also been interpreted as fragments of the great ore-body; and from this it has been concluded that the porphyry of the hanging-wall should be younger than the mass of ore.

The unmistakable "consanguinity" between the soda-syenite and the porphyries is manifested by the high percentage of Na, which varies between 5.5 and 7.5 per cent.; apatite, titanite, and magnetite are, besides, minerals common to the syenite, the porphyries, and the ore-deposits.

The Ores.—(In the following exposition of the Kiirunavaara and Luossavaara ore-deposits, I follow chiefly the official report made by Hj. Lundbohm in 1898.) The iron-ore occurring among the porphyry-masses forms, on the whole, pure, nearly homogeneous ore-deposits; other minerals found in it are of comparatively subordinate significance.

A property characteristic of the ore of Kiiruna-, Luossa-, and Tuollavaara is its general extremely fine-grained texture, which proves that it has been subject, to a slight degree only, to the action of recrystallizing agents. By this structure, which is also found in a few other ores in Norrbotten, this ore-type is distinguished from the rest of the Scandinavian ores.

The only mineral that occurs in the ore with undoubtedly primary characters is apatite, the distribution of which is exceedingly irregular, so that the percentage of phosphorus in the ore varies greatly.

In Kiirunavaara, chiefly close to the foot-wall, but also here and there in the interior of the ore, occurs an ore-type with mostly grayish-black and dull, compact fracture (Lundbohm's type 5). When examined with the microscope it proves to be interlarded with apatite individuals idiomorphically developed; its phosphorus-percentage is from 3 to 6 per cent. This ore frequently presents a stratiform structure.

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The ore-type which quantitatively predominates contains the apatite in nodules and irregular lenses (Lundbohm's type 4). Here, too, the apatite seems to be, at least in part, of primary origin, since it occurs partly as a minute impregnation of the ore, partly in irregular nests and veins, giving rise to a struc-

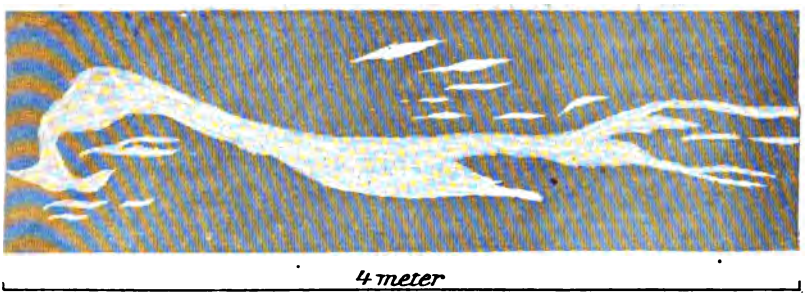

FIG. 10.—IRREGULAR VEINS OF APATITE IN MAGNETITE, KIIRUNAVAARA (Lundbohm).

ture which bears some resemblance to a largely developed flow structure (Fig. 10).

Primary structural forms which may be referred to flow-structure may also be observed in the relation between different ore-types, when, *e. g.*, one type contains fragments or "*schlieren*" of another type, or when one ore-type occurs as intruding dikes

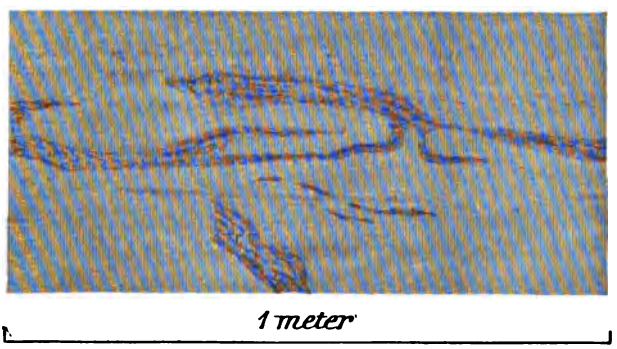

FIG. 11.—FLOW-STRUCTURE IN MAGNETITE, KIIRUNAVAARA (Lundbohm).

in another (Fig. 11). Especially on weathered ore-surfaces these structures are easily observable.

It is likely that the highly phosphoric ores, though embracing several types, must in general be regarded as primary, and those poorer in apatite as secondary, leached, and in part re-

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crystallized. The latter also contain martite and specular hematite, the occurrence of which is here evidently secondary.

In the low-phosphorus ores occur also calcite (formed at the expense of the apatite), secondary quartz in fissures, and secondary silicates as amphibole, talc and chlorite minerals. The secondary ore-types are sometimes porous, the more soluble minerals having been leached out.

From a practical point of view, the ores in question have been divided into several classes according to their percentage of phosphorus. Those classes which range above 1 per cent. constitute the principal mass of the ores of Kiirunavaara. "As a fairly certain result of the examinations of the ores with regard to their percentage of phosphorus, it may be said that ores containing less than 0.05 per cent., and ores with from 0.05 to 0.1 per cent., of phosphorus, occur separately in such a mode that they can be utilized, but that both kinds, especially the former, are, as regards quantity, rather subordinate to those richer in phosphorus. The main mass of the ore of Kiirunavaara contains more than 0.8 per cent., generally from 1 to 2 per cent., not infrequently from 3 to 4 per cent. or still more, of phosphorus." [15] "On the whole the iron ores of Kiirunavaara may be said to contain a higher percentage of phosphorus than any other known ore-deposit of great extent." [16]

The phosphorus-percentages of the ores of Luossavaara, although exceedingly variable, seem mostly to be less than 0.1 per cent. In Tuollavaara, a part of the ores are also remarkably free from apatite, showing a phosphorus-percentage amounting to only hundredths of 1 per cent., but also here ores high in phosphorus occur.

Owing to the absence of other impurities, the iron-percentage of these ores is very high. That of Luossavaara fully equals or even exceeds that of the ore of Kiirunavaara. Apart from the ores richest in apatite, the average amount of iron is from 68 to 69 per cent.

The Forms and Dimensions of the Ore-Bodies.—The large ore-deposit of Kiirunavaara is an enormous mass, roughly rectilinear in outline. In length, the ore is exposed in the hills for

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[15] Hj. Lundbohm, *Kiirunavaara och Luossavaara Jernmalmsfält I Norrbottens Lan* (1898), p. 45.

[16] Lundbohm, *loc. cit.*, p. 55.

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more than 2,800 m., and through drilling and magnetical investigation has been proved to extend at least 1,200 m. further. In a northern direction below the level of Lake Luossajärvi, the deposit seems to divide in two narrow parallels, only known through magnetic investigation. The width of the deposits in some places exceeds 200 meters.

This enormous ore-body is cut in only two places by narrow masses of porphyry, extending from the hanging- to the foot-wall.

The boundaries of the ore against the rock are rather irregular, as may be seen in the outcrops, and the width of the deposit is thus very variable. Also the depth of the ore-body is remarkably irregular, which makes any calculation of the ore-quantity uncertain. The dip varies between 45° and 60° east. Generally, the dip of the foot-wall is a few degrees greater than that of the hanging-hall, indicating a downward decrease of width.

By diamond-drilling the dimensions of the ore-body have been in the main determined to a depth corresponding to the level of the Luossajärvi lake; beneath this level the ore is in some places proved to extend to a depth of 200 m., but concerning the dimensions of the body at this depth nothing is known.

The Luossavaara deposit exhibits the same features as Kiirunavaara, only on a smaller scale. It is known for a length of at least 1,200 m., of which 750 m. is exposed. This ore-body dips east about 65°. The decreasing width towards the depth is here still more evident than at Kiirunavaara.

Contact-Zones.—As to the relation of the iron-ore to the surrounding rocks, in many places a contact-zone, one meter or a few meters in breadth, is observable, in which occurs a mixture of ore and minerals belonging to the porphyry.

Thus in Kiirunavaara, within the porphyry of the foot-wall, ore-veins with indeterminate boundaries are found running through the rock. The branching veins form a net of wide meshes inclosing pieces of the rock. In one direction the ore-veins pass into fine fissures in the almost unaltered porphyry; in the other direction the magnetite prevails, and angular, strongly uralitized fragments of porphyry are found in the compact ore-mass. At last one finds only a dark-

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green amphibole in the magnetite as a reminder of the silicate-rock.

Inclosed masses of the surrounding rocks occur in many places in the ore, especially in the neighborhood of the foot-wall. Thus, in one place within an area of 40 m. in breadth streaks of, in part, considerably altered porphyry alternate with streaks of iron-ore; remains of porphyry altered into amphibolite and chlorite have also been found in the ore.

Also in Luossavaara similar contact-zones are found, though less accessible to the observer, on account of the thick covering of glacial deposits. Through the porphyry of the foot-wall, west of the top of the mountain, run veins up to 1.2 m. in width, containing magnetite, titanite and amphibole.

In Tuollavaara, also, a brecciated structure is found, perhaps better observable, though occurring on a smaller scale than in Kiirunavaara. The surrounding rock is irregularly cut by magnetite-veins and veinlets, giving rise to a breccia of sharp-edged porphyry-fragments, cemented together by magnetite; this phenomenon decreases as the distance from the ore-limit increases. Isolated breccia-like pieces of porphyry also occur inclosed in the iron-ore.

The Genesis of the Deposits.—The genetic connection of these ores with the porphyry-rocks is so manifest, that it has been admitted by all who have expressed their opinion on the subject. Even those geologists (Hummel and Gumælius in 1875, Fredholm in 1891) who regarded the porphyries as sedimentary *hälleflintor*, acknowledged this connection, and consequently considered the ores as sedimentary formations. Löfstrand, in 1891 and 1892, in describing other basic segregations and vein-like formations of iron-ore in acid igneous rocks, pointed out that the connection of the ores of Kiiruna with the porphyries ought to be interpreted in the same way. The same opinion was expressed more positively in 1898 by Högbom,[17] who laid special stress on the agreement with the deposits connected with syenitic rocks in the eastern Ural: Wyssokaia Gora, Lebiajaia, and Gora Blagodat. A similar opinion was pronounced at a later time by O. Stutzer, who holds that the ores have been formed in an epigenetic-magmatic way as "eine nach oben

[17] *Geologiska Föreningens Förhandlingar*, Stockholm, vol. xx., p. 115 (1898.)

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gewanderte magmatische Ausscheidung"—*i. e.*, a magmatic vein-formation.

A pneumatolytic sedimentary mode of formation has been maintained by Bäckström and, later, by De Launay (1903). The latter author, who gives the most detailed exposition of this view, has formed the following conception of the process: The porphyry of the foot-wall is an effusive rock, on which the iron-ore, formed through the decomposition of chloride and sulphide of iron in contact with water, has been deposited. Later on a new eruption of porphyry followed, by which the porphyry of the hanging-wall was formed.

This interpretation is based on the opinion that the ore is younger than the porphyry of the foot-wall and older than the porphyry of the hanging-wall, which, however, is hardly compatible with the fact that the magnetite is, in places, completely surrounded by the basic porphyry.

The above-mentioned fluidic structures in the magnetite can be accounted for only by assuming that the magnetite, together with the greater part of the apatite, has formed a segregation from an iron-alkali-silicate-magma, intruded as a vein between the porphyries. After this intrusion the effects of pneumatolytic agencies, which are especially well-marked at the contact with the basic porphyry, have arisen. Högbom has given a theoretical exposition of the formation of ores of this kind. He starts from an iron-alkali-silicate-magma composed, approximately, in the proportion of 1 molecule of orthoclase, 1 molecule of albite, and 1 molecule of magnetite. Such a magma differs in composition from known and common magma-types only by containing a little more ferric oxide and a somewhat smaller amount of lime and magnesia. As, at the solidification of such a magma, the larger part of the iron must segregate as magnetite, because, owing to the absence of lime and magnesia, it cannot combine with the silica, the differentiation of two rocks, one chiefly consisting of magnetite, the other of feldspar, is easily accounted for. Högbom, therefore, holds that the alkali-silicate-magmas rich in iron, to which petrography has as yet paid but little attention, have a just claim to a place in the system, and that their most typical representatives are magnetite-bearing syenitic rocks of this kind.

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Mertainen and Painirova.

An interesting illustrative complement of what is known about Kiiruna is afforded by the statements, scanty and incomplete though they are, which are accessible concerning Mertainen, an ore-deposit of comparatively little importance, situated about 30 km. SE. of Kiiruna, within the group of deposits including also Svappavare and Leveäniemi, from which, however, it is quite different as regards geological conditions. These ores, too, are connected with a syenite-porphyry, mainly consisting of a soda-feldspar. At the contact with the deposit this rock has undergone a more or less advanced transformation of pneumatolytic character. The original basic rock-constituents have altogether disappeared, and new formations of magnesia-silicates appear in their place. The soda-feldspar has, in part, been transformed into scapolite, but also biotite and titanite have been produced.

In the syenite-porphyry, magnetite occurs, partly finely disseminated, partly as small segregations from the size of an almond to that of an egg. These have also been interpreted as cavity-fillings. The deposit proper, however, consists of a magnetite breccia (Fig. 12); a fine-grained magnetite fills the corrosion-fissures of the rock, sometimes associated with amphibole, less often with apatite.

Several of the ore-bodies have been found to extend to no great depth. The ore is a very rich magnetite, with a low percentage (generally less than 0.5) of phosphorus.

A deposit of quite the same character, Painirova, occurs 8 km. south of Mertainen. Here, too, one meets with a porphyry breccia with magnetite-veins and irregular ore-nodules; some of the veins contain apatite in abundance. The deposit seems to be of theoretical interest only. The syenite-porphyry presents partly a remarkable stratiform structure, and the gradation from the massive porphyry into this structure may be followed step by step.

The deposits of Mertainen and Painirova are evidently, like those of Kiiruna, genetically connected with the syenite-porphyries. But while at Kiiruna the magmatic characters of the ore-mass are most marked and the pneumatolytic characters subordinate, at Mertainen the reverse is the case. Breccia-for-

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mations due to corrosion occur in both places, but form at Mertainen the larger portion of the deposit. The pneumatolytic phenomena which, at Mertainen, are manifested by a far-advanced scapolitization of the feldspar of the porphyritic rock

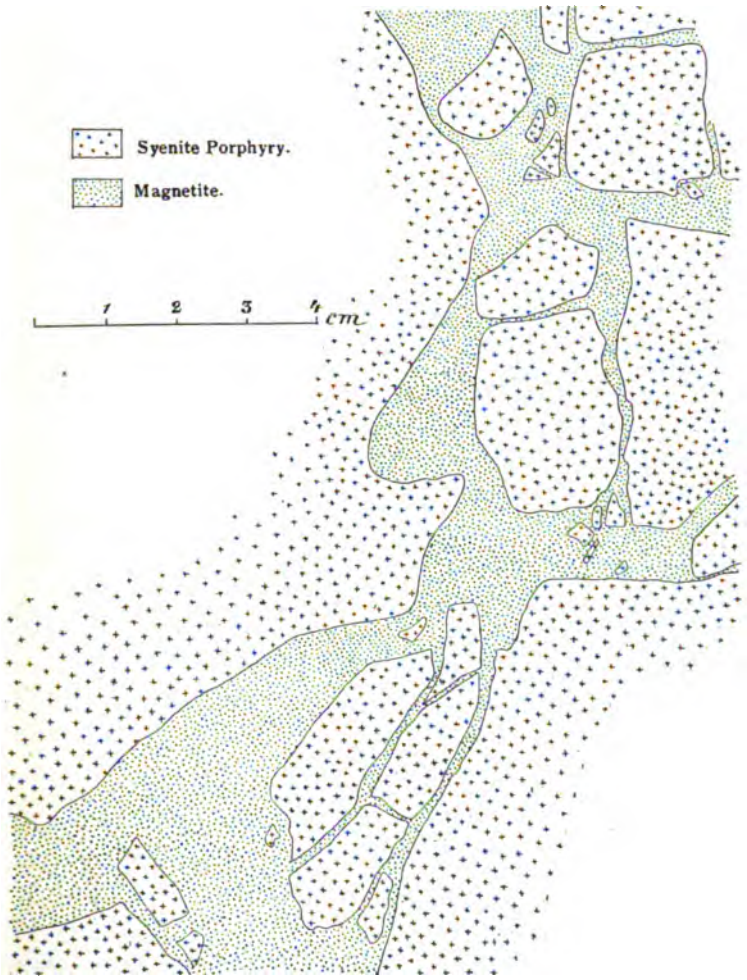

FIG. 12.—BRECCIATED IRON-ORE FROM MERTAINEN (W. Petersson).

and by the formation of biotite and titanite, are at Kiiruna of minor importance. On the other hand, Mertainen presents nothing analogous to the pure ore-masses of magmatic origin, and partly of fluidic structure, which form the main deposit at Kiiruna.

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Ekströmsberg.

Ekströmsberg is situated about 30 km. west of Kiirunavaara. Here, too, the ore-deposit is connected with porphyry-rocks of syenitic composition; in the neighborhood, however, occur granites as well as greenstones, probably belonging to the same eruptive series. Part of the rocks show strong traces of dynamic action. The rock on both sides of the deposit is a quartz-porphyry of about the same acidity as the quartz-keratophyre of Kiiruna, yet differing from it by being a marked potash-rock. The deposit consists of a complex of magnetite and specular hematite, cut longitudinally by intrusive porphyries and porphyrites.

Though differing, by the potash of the surrounding porphyry-rock, from Kiiruna and its soda-syenite-porphyries, the deposit of Ekströmsberg belongs, in other respects, to the same well-defined geological group.

Besides the aforesaid deposits, a great many other iron-ores are known in Norrbotten. These ores are either associated with syenitic rocks, having then frequently a brecciated structure, or occur in the crystalline schists, sometimes as impregnations of magnetite, of great extent but little concentrated. Owing to the thick covering of moraine, these deposits have, as yet, been but partly and imperfectly examined; several of them are known only through their effect on the magnetic needle. Most of them seem to be of little practical importance.

Analogous Deposits.

The ores in Norrbotten of the Kiiruna type belong to a particularly well-defined geological-petrographical type, which is also met with in other parts of the world.

Högbom has already, in the above-mentioned paper, pointed out the agreement between the iron-ore deposits in the eastern Ural and the Kiiruna type. In the iron-mountains of the Ural a secondary epidotization of the rocks has taken place on a larger scale than in Norrbotten, especially along certain planes of dislocation. On the other hand, the pneumatolytic characters so well marked in the Kiiruna type are not met with in the deposits of the Ural. The secondary transformations, such as the development of martite and specular hematite, the leaching

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out of pyrite and apatite, the formation of porous ore or ore containing calcite, and the accumulation of the apatite chiefly near the foot-wall, are common to the two districts.

The ores of the (for the most part exhausted) deposits of Iron Mountain and Pilot Knob in Missouri, which also occur in association with porphyry-rocks, have been compared to and classed with the Kiiruna type by several authors. At Iron Mountain the ore mined occurred as veins and irregular masses of martite and specular iron-ore in a mostly decomposed porphyry of Archæan age. At Shepherd Mountain similar deposits in porphyry were worked. The deposits of Pilot Knob, on the contrary, are secondary redepositions of the primary iron of the porphyry; they seem to bear a strong resemblance to the deposit of specular iron-ore in the Hauki schists east of Luossavaara.

Also, the Mexican deposits at Durango and Las Truchas agree in some respects with those in Norrbotten; but their geological conditions have not, as yet, been sufficiently investigated to make a direct comparison possible.

Group III.—Iron-Ores Formed by Magmatic Segregation in Basic Eruptives.

The ores of this kind form a natural and well-defined class encountered in all parts of the world. That they are genetically connected with eruptive rocks has long been admitted. The nature of their facies of differentiation was not understood until the differentiation of rock-magmas was clearly conceived. In this regard, their structural characters, which are the same as those of the eruptive rocks, and their frequent presentation of all degrees of transition to the normal rock are evidential. (Fig. 13.)

Magmatic differentiations of this kind seem to be connected only with intrusive eruptives, and occur in laccolites as well as in vein-like intrusions. In general, a distinction can be made between such differentiations as have taken place within the mass of the laccolite, *in situ*, and such as have taken place in the deep-seated magma. In the latter case the product of magmatic segregation has been carried up to the level of the laccolite by a separate act of eruption.

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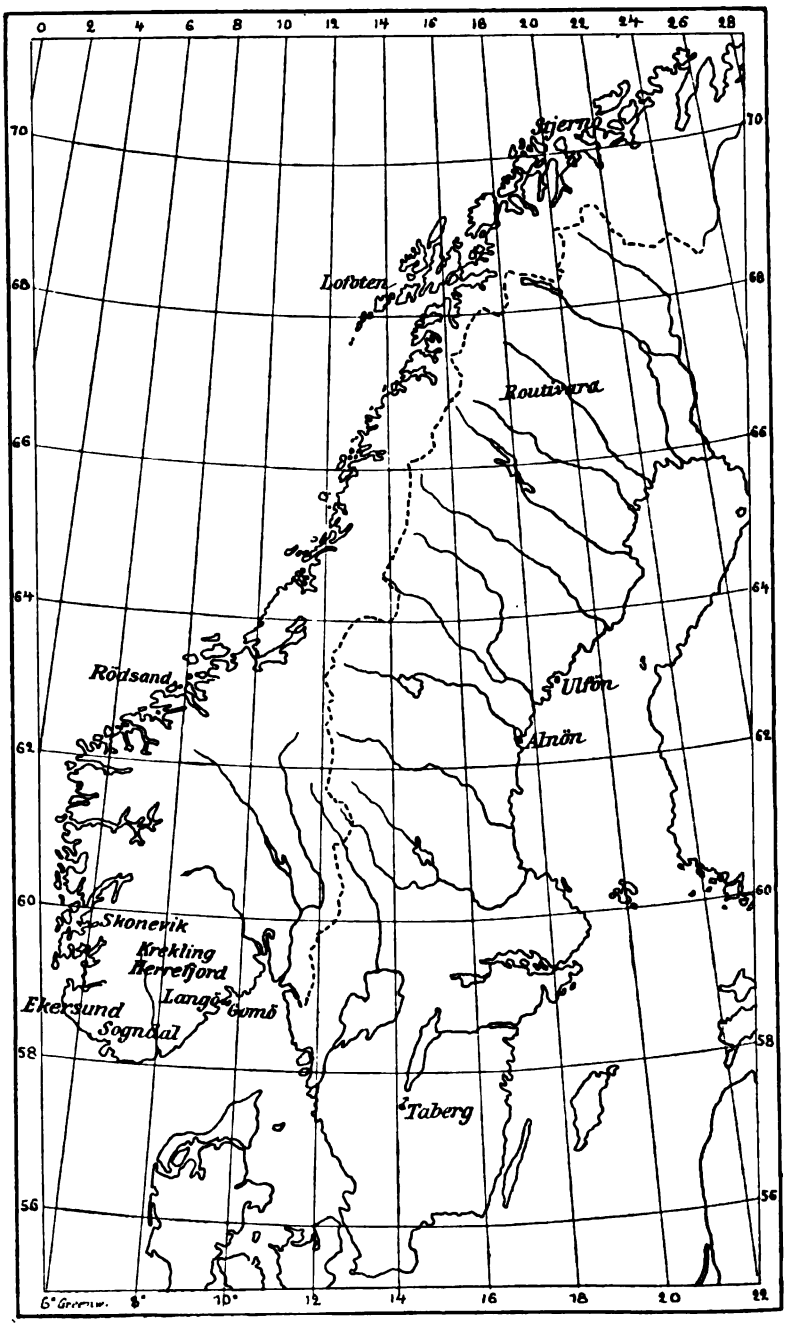

FIG. 13.—DISTRIBUTION OF TITANIFEROUS IRON-ORES.

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Taberg in Smdland.

As far as a hundred years back Hausmann[18] expressed the opinion that "the mass of Taberg is a greenstone bed of tolerably great thickness, mixed with much iron-ore and lying in gneiss." Through the investigations of A. Sjögren[19] and Törnebohm[20] it was established that the ore-deposit of Taberg ought to be considered as a segregation in a basic eruptive, the chief constituents of which are olivine, plagioclase, a rhombic pyroxene and magnetite. The structure is that of a deep-seated rock, and the rock, which has been called hyperite by the Swedish geologists, is olivine-norite according to the nomenclature of Rosenbusch. Taberg was the first iron-ore deposit interpreted as a phase of an eruptive rock. Törnebohm[21] says that the Taberg ore "may be regarded as a variety of hyperite rich in iron." As the ideas of magmatic differentiation were not clearly formulated until later, the nature of the ore could not in 1881 be expressed in plainer terms. The whole of the eruptive constitutes an intrusion (laccolite) in the surrounding gneiss, above which it now rises, by reason of its greater power of resistance to erosion. (Fig. 14.)

The ore-deposit occupies the central portion of the mountain. The ore-segregation consists of titaniferous magnetite and olivine and has received the petrographical name magnetite-olivinite. Where it approaches the normal rock, it first takes up plagioclase, then pyroxene, so that there is a transition from the ore to the normal rock, which also contains magnetite and olivine. The ore-stock thus forming the kernel of the mountain, is next surrounded by a shell or mantle of normal gabbro, which, in its turn, towards the inclosing gneiss, passes into the schistose, dynamo-metamorphic border-facies, the gabbro-amphibolite.

The ore is poor throughout, carrying generally from 20 to 30 per cent. of iron. Vein-like segregations containing up to 60 per cent. occur as rare exceptions. Titanic acid varies between 4 and 6 per cent.; the percentage of phosphorus does not exceed 0.1; to which may be added a constant percentage of vanadine.

[18] *Reise durch Skandinavien*, 1806-07, pt. i., pp. 158 to 167.
[19] *Geologiska Föreningens Förhandlingar*, vol. iii., p. 42 (1876).
[20] *Geologiska Föreningens Förhandlingar*, vol. v., p. 610 (1881).
[21] *Loc. cit.*

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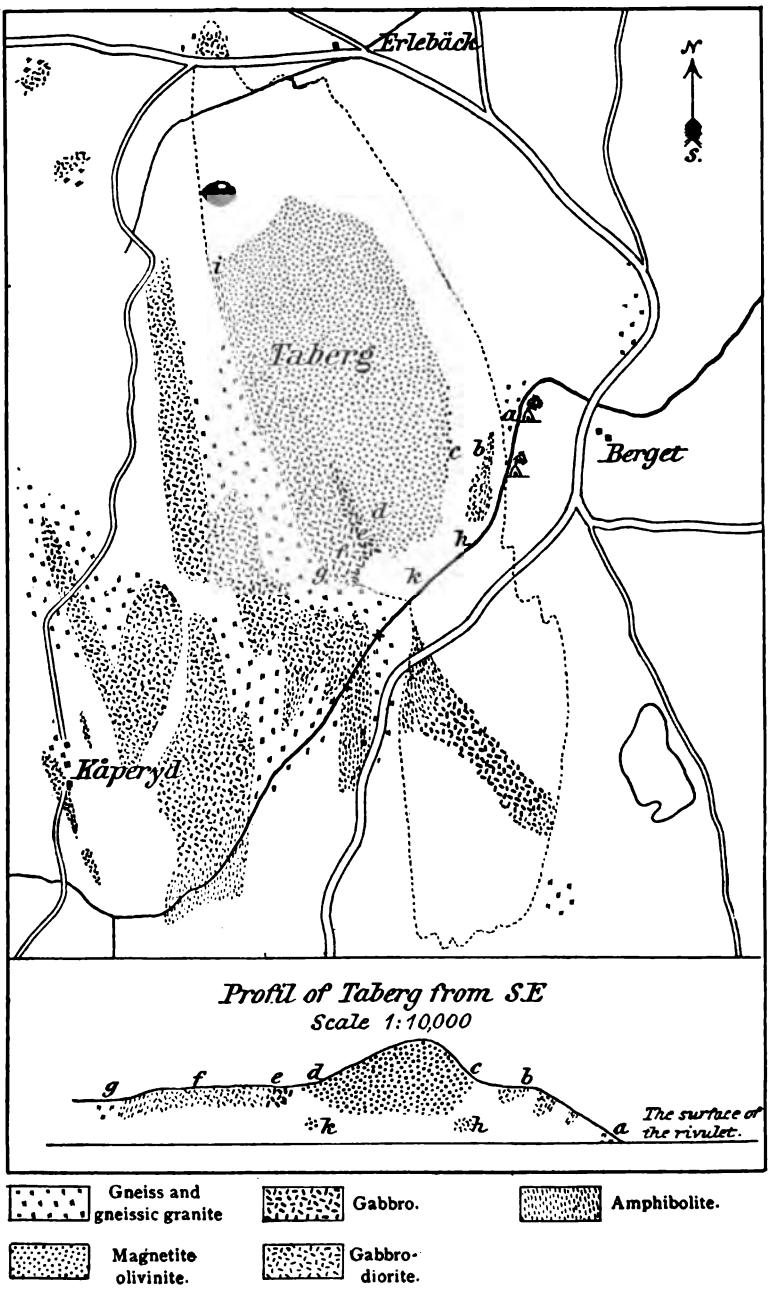

FIG. 14.—TABERG (Törnebohm).

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Vanadine was discovered in 1830 by Sefström in ingot- and bar-iron produced from Taberg ore.

The ore of Taberg has been mined since time immemorial; during the 17th and 18th centuries it was the raw-material of a local iron-manufacture. As late as the fifties of last century twelve, and in 1875 ten, small furnaces were in operation, which used, chiefly at least, this ore. Since 1890, mining has been confined to the work required by law to escape forfeiting the licenses.

Other minor deposits of the same kind as Taberg occur in several places of central and southern Sweden. At Inglamåla in Jönköping a similar deposit has been worked, the ore being mixed for use with Taberg and lake ores.

Of essentially the same kind as these Swedish deposits are also several deposits in Norway. According to Vogt's description of the gabbro-masses in Langö and Gomö near Kragerö, these deposits agree very nearly with that of Taberg. They occur in an "olivine-hyperite," and the gradual concentrations can be followed from a gabbro rich in iron to a darker facies, poorer in plagioclase but richer in magnetite, at last passing into a rock containing no feldspar, consisting of magnetite and ilmenite together with iron-magnesia-silicates. The quantity of this ore is considerable, and at times it has been mined; it contains about 40 per cent. of iron with from 7 to 10 per cent. of titanic acid. Similar to these is a minor deposit at Herrefjord near Porsgrund; of a somewhat different character is the deposit at Krekling, 6 km. E. of Kongsberg. The latter is distinguished by an uncommonly strong concentration of apatite in connection with the iron-segregation. The ore, consisting of concentrations of titano-magnetite, iron-magnesia-silicate and apatite (sometimes amounting to 25 per cent.), occurs in a gabbro containing hypersthenite.

To this group we may probably also refer the numerous smaller deposits of more or less titaniferous ores which occur in many places in the older Archæan formation, in Sweden as well as in Norway, in diorites, amphibolites and amphibolite-schists, which rocks are to be interpreted as dynamo-metamorphosed eruptives. Such dynamo-metamorphosed basic eruptives with iron-ores are known from Rödsand, Tingvoldsfjord in Romsdal, where the deposit is said to lie in red and

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gray gneiss; the iron-ore is titano-magnetite containing 8.20 per cent. of TiO_2 and 52 per cent. of Fe, mixed with amphibole and a little garnet. Of similar character is a deposit at Stalkjärn near Egeland, in Nedenäs.

Segregations in Diabase Younger than the Dala-Sandstone.

In the small islands scattered along the coast of Angermanland occurs a diabase in large masses overlying the Algonkian quartzite formation. The diabase forms bed-like intrusions between the stratified quartzites, and has been laid bare by erosion. It is an olivine-diabase of the type to which Törne-

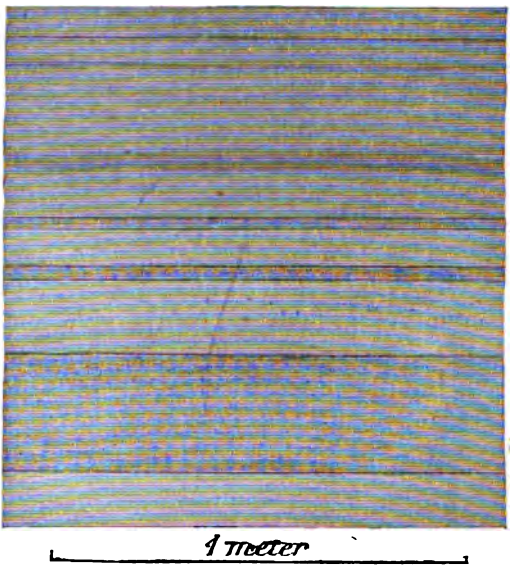

FIG. 15.—BANDED IRON-ORE FROM ULFÖ (H. Lundbohm).

bohm has given the name Asby diabase. The structure of this rock is ophitic; frequently a remarkable alternation of light and dark bands (leucocratic and melanocratic beds) is observable; in the melanocratic bands a concentration of titano-magnetite has sometimes taken place to a degree that gives them the character of iron-ore. This is the case in the islands of Ulfö, where a horizontal bed from 0.3 to 0.5 m. in thickness and of fairly large extent has been worked. (Fig. 15.) Overlying and underlying this bed there are several thinner ore-bands presenting a regularity like that of stratified rocks. It is evident that the distribution into bands which the magnetite

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shows is due to the movement during the injection of the magma, and that the differentiation has taken place in the deep-seated magma basin. (The deposits in Ulfö bear great resemblance to the banded gabbro in the isle of Skye, described by Harker, and that of Deneschkin-Kamen in the northern Ural, described by Loewinson-Lessing.)

The percentage of titanic acid may amount to 25 per cent. In the island of Trysunda the segregations occur in more irregular masses (Fig. 16). These ores are known since the middle of the 18th century, and have at times been utilized on a small scale—the quantity mined seems never to have exceeded 1,000 tons in the year—for the small blast-furnaces in the neighbor-

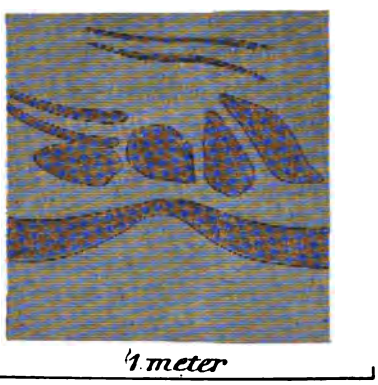

FIG. 16.—SEGREGATIONS OF IRON-ORE, ISLAND OF TRYSUNDA (Lundbohm.)

hood, in which they have been mixed with other ores. Since 1876 no work has taken place.

Iron-Ore in Nepheline-Syenite.

In the island of Alnö, off Sundvall, occurs a boss of nepheline-syenite investigated and described by Högbom. Dike-formed rock-facies of the same eruptive series run through the Asby diabase; the syenite-rocks, consequently, are also younger than the sandstone which underlies the diabase. The rock presents within the mass, only about 4 km. in extent, a great many varying differentiated facies, with transitions from one to another. Also, limestones occur, which have been interpreted as segregations of an ultra-basic magma. In places the titano-magnetite has concentrated to poor ores, mostly, however, only

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impregnations. Like the rock itself, these concentrations are of highly varying composition; some of them show a large amount of apatite with a P-content of 7 per cent.; others are free from phosphorus. The concentration of titano-magnetite has been accompanied with a concentration of olivine; on the other hand, feldspar is absent from the basic differentiations, which is also frequently the case with titanite, nepheline, and cancrinite. For the rest, the ore is mostly accompanied by pyroxenes. The percentage of titanic acid also varies from 9 to 12 per cent. These ore-deposits were worked in several small mines during the middle and latter part of the 18th century. Abandoned towards the end of that century, the mines were again worked in the beginning of the eighties of last century, but were abandoned again in 1884. On account of their high percentage of titanium, the frequently high amount of phosphorus and the low iron-percentage, a profitable exploitation of these ores is not very likely.

Ekersund and Soggendal in Norway.

In these localities the ore-deposits occur in a district of igneous rocks extending over 1,450 sq. km., and forming a section of a colossal laccolite consisting of a series of highly differentiated basic rocks of eugranitic structure. The age of the intrusion is considered to be post-Silurian. The most important differentiation-facies are: a feldspar rock consisting chiefly of labradorite, which occupies about two-thirds of the area; norites; and augite-granites; the first-mentioned rock being the oldest. Dikes of pegmatitic norite occur, besides dikes of norite of normal grain and of olivine-diabase. In the labradorite-rock as well as in the norite occur differentiation-facies rich in ilmenite, partly as ilmenite-norite, partly as veins of pure ilmenite. (Fig. 17.)

The deposits are distributed in two areas, the southernmost of which is the Soggendal field, where the main deposit is Storgångeu, a dike of ilmenite-norite, which, with sharply-defined boundaries, traverses the labradorite-rock; its length is over 4.6 km., the width varies between 20 and 70 m.; it contains from 40 to 80 per cent. of titanic iron, corresponding to from 20 to 40 per cent. Fe. The rock is often banded along the borders of the dike in alternating melanocratic and leuco-

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cratic *schlieren*, indicating a differentiation in the deep-seated magma. It sometimes contains sharp-edged fragments of the wall-rock. In Blåfjeld, irregular segregations up to 50 m. in extent, of nearly pure titanic iron, are met with. (Fig. 18.) According to Kolderup these deposits are in part real intrusive dikes, younger than the surrounding labradorite-rock, in part *schlieren*, lenses and dike-shaped masses formed by magmatic

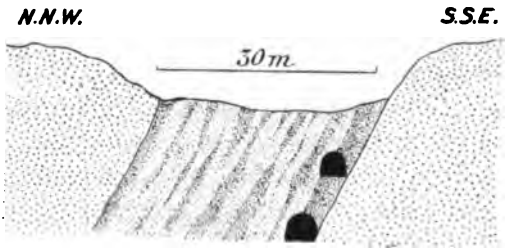

FIG. 17.—PROFILE OF STORGÅNGEN, SOGGENDAL (Vogt).

concentration *in situ*. The former (type Storgången) owe their origin to a differentiation of the deep-seated magma, the latter (type Blåfjeld) to a laccolitic differentiation. Kolderup describes ten such deposits, some of which occur as segregations, some as dikes. They are met with in the labradorite-rock as well as in the norite; the ilmenite dikes are consequently younger than these two rocks.

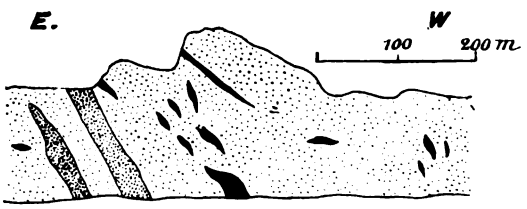

FIG. 18.—BLÅFJELD, SOGGENDAL (Vogt).

In the Ekersund field, dikes of titanic iron of very irregular character occur distributed along a large dike of olivine-diabase. The ore-deposits, from 50 to 400 m. in length and up to 40 m. in breadth (only two of them, however, reach the latter dimension), frequently contain pure titanic iron. As these deposits show sharply-defined boundary-lines at the contact with the inclosing rock, and also contain fragments of it, it can hardly be doubted that they are true dikes.

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The ores of this district have been worked at different periods. Every attempt at a profitable exploitation has, however, proved a failure, owing to the high percentage of titanic acid and the low percentage of iron. On an average, they may be said to contain 40 per cent. Fe. and from 40 to 42 per cent. of titanic acid.

In the Soggendal field a good deal of mining-work was performed from 1864 to 1876, especially in Storgången and Blåfjeld. Also in the Ekersund field a considerable quantity of ore was mined from 1870 to 1875, and the total export from the two fields from 1864 to 1876 amounted to about 90,000 tons of ore, with an average percentage of 40 per cent. of iron and 40 per cent. of titanic acid. These ores were mined by the Norwegian Titanic Iron Co., and were smelted at the works of the company at Norton, near Stockton-on-Tees, England. In 1899 and 1900 a few thousand tons were extracted from the Soggendal mines.

Deposits of the same type as Ekersund and Soggendal are known in a great many places along the coast of Norway. The abrasion by the Atlantic and the deep indentation of the fjords running far inland have laid bare a large number of similar laccolites. Kolderup describes such deposits in the labradorite-rock field of the peninsula of Bergen, where minor deposits occur in groups at several places. At Bogstö in Skonevik, the peninsula of Folgefonden, such segregations occur in gabbro. The age of the rock and its geological correspondence to the aforesaid eruptive series is, however, uncertain.

Routivare.

One of the largest deposits of this class in Scandinavia is Routivare in Norrbotten. This deposit, which has been known for a long time, was formed by segregations in an intrusive rock occurring as a laccolite in the metamorphic Silurian formations. The strongly dynamo-metamorphosed character of the rock may possibly be explained as depending on its occurrence in the overlap-zone running along the eastern slope of the Scandinavian mountain-range. The ore-area has been estimated at 300,000 sq. m. at the surface. The eruptive rock consists of a gabbro altered almost past recognition and strongly saussuritized. The primary basic minerals have

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totally disappeared; sometimes secondary aggregations of amphibole, serpentine, and garnet occur. The plagioclase is mostly recrystallized. The structure generally shows a lamination due to crystallization. The chemical composition agrees, in the main mass of the rock, with that of a labradorite rock. Besides this rock there occur in places, partly accompanying the ore, segregations rich in Mg and a composition corresponding to that of an olivine rock. The iron-ore, which occurs in highly irregular segregations, traversing the rock irregularly and with sharply-defined bounding surfaces, consists chiefly of titaniferous magnetite, but also contains ilmenite; other minerals met with are spinel, olivine, and pyroxene. This ore carries, in round numbers, 50 per cent. of iron, and 10 per cent., or more, of titanic acid.

The deposit at Vallatj, a few kilometers north of Routivare, is of similar character, but of smaller extent, with an ore-area only one-eighth or one-tenth of that of Routivare.

Summary.

Though the rocks which inclose the deposits of this class differ in composition (as real gabbros, norites, diabases or nepheline-syenites), as well as in age (some of them belonging to the oldest Archæan formation, and the youngest dating from post-Silurian time), yet a marked degree of basicity seems to be a necessary condition for the formation of such concentrations. Vogt puts the highest acidity at 57 per cent. SiO_2. The majority of the deposits occur in rocks with a silica-percentage of from 48 to 54 per cent. The different facies of gabbro-rocks inclose deposits of different character. The pure ilmenite-segregations (Ekersund, Soggendal, Lofoten, etc.), seem to be confined to the labradoritic rock. In the same rock-series we also find the ilmenite-norite. In the olivine-gabbro, rich in magnesia, which incloses the deposit of Taberg, the segregation has assumed the mineralogical character of magnetite-olivinite, and in the rocks which are richer in alumina, magnetite-spinellite has been segregated, as at Routivare, Andopen, on Stjernö in Finnmarken, and elsewhere.

A remarkable feature of the ore-concentrations in question is their occurrence, almost without exception, in the central parts of the eruptive masses. This gives them a character dif-

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ferent from that of those concentrations of basic constituents in an eruptive rock which are often met with along the margins of rock-veins, and in which the enrichment has not proceeded so far as to form an iron-ore.

As to the degree of concentration of the iron, the ore-types differ also from one another. The concentrates richest in iron are the magnetite-spinellites, with an iron-percentage exceeding 50 per cent. (Routivare, 50 to 54; Solnör, 54; Hellevig, 51; Andopen, nearly 60). Next to these come the segregations of the nepheline-syenites, of which in Alnö some contain from 46 to 53 per cent.; and after these the ilmenite-segregations in the labradoritic rock and the norite, with about 40 per cent. of iron, and nearly as high a percentage of titanic acid. Poorer still are the ores of the olivine-gabbros of the Taberg type, which contain little more than 30 per cent. of iron. Comparable to them are the ores of the olivine-diabase (the Ulfö type), with about 34 per cent. At the bottom of the scale stands the ilmenite-norite, the Storgång type, with about 21 per cent. of iron.

Besides the iron, it is chiefly the titanic acid, the magnesia, and the alumina that have been concentrated. The percentage of titanic acid is highest in the ilmenite-segregations of the labradorite and norite, where it amounts to from 39 to 43 per cent.; next come the ilmenite-norites, which contain 18 per cent.; the magnetite-spinellites (type Routivare) vary between 10 and 18 per cent.; the segregations of the nepheline-syenite show about 10 per cent. (the Trygg mine, in Alnö, from 9.10 to 12.14); in the Taberg type, the magnetite-olivinite, the percentage of titanium, like that of iron, is the lowest—viz., 6.30 in the Taberg ore and 8.50 in the ore of Långhult. The Ulfö type, with about 10 per cent. TiO_2, shows the greatest agreement with the Taberg type.

The concentration of magnesia has taken place not so much in the ore as in the concentration-facies between the normal rock and the segregations richest in iron. It manifests itself in the formation of Mg-Fe-silicates of the olivine- and pyroxene-groups. The rock-facies, which have received the names magnetite-olivinite, ilmenite-norite, and ilmenite-enstatite, have originated in this way. A certain percentage of Mg is found even in the purest segregations of ilmenite in the labradoritic

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rock, owing to a mixture with $MgTiO_3$. The alumina left in the final concentrates combines particularly with the magnesia, thus forming spinel, the formation of which is favored by the relation-deficiency of silica in the magma. This mineral occurs in the following ore-types: the magnetite-olivinite (Ransberg); the ilmenite-norite (Ekersund); and the magnetite-spinellite (Routivare, Hellevig, Lofoten, and Stjernö). Vogt has called attention to the fact that the Mg-percentage increases in the first stage of concentration and then diminishes. While the original magma contains more Al_2O_3 than MgO, the case is quite the reverse in the earlier stages of concentration. In the final product, however, the amount of alumina again exceeds that of magnesia.

Besides the aforesaid substances, chrome and vanadium, which occur in small quantities, have undergone a concentration. The phosphorus, on the other hand, is not in general concentrated to any noteworthy degree. To this, however, there are exceptions, such as the segregations of ilmenite-norite traversing the labradoritic rock in the Soggendal-Ekersund field, which contain a fairly high percentage of phosphorus, while the surrounding rock carries little, and the above-mentioned concentrations, rich in apatite, in the hypersthene-gabbro at Krekling, Norway, and also in the nepheline-syenite of Alnö.

That silica, lime and alkalies occur in smaller quantities in the concentrates than in the rest of the rock-mass is manifested mineralogically by the total absence of feldspar from these concentrations.

Vogt has pointed out[22] that in several places in Lofoten and Vesteraalen, in the labradoritic rock containing olivine and hypersthene, there are besides segregations of magnetite-dialagite, also *schlieren*-like segregations of pure olivine rock as well as of hypersthenite, This shows that in the same magma differentiation processes following different lines have taken place nearly contemporaneously. As extreme basic segregations the limestones occurring in the nepheline-syenite may also be explained.

Many different attempts to suggest the cause of these differ-

[22] *Zeitschrift für praktische Geologie*, vol. xiv., pp. 217 to 233 (1906).

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entiations have been made, but no satisfactory explanation has as yet been proposed.[23] It seems as if, with respect to this kind of segregations, the view according to which the magma is regarded as a mixture of different liquids, partly insoluble in one another, were decidedly preferable to the theory which considers the laws of dilute solutions applicable to the magma. The principle of limited solubility must be considered as the physico-chemical principle governing the differentiation-phenomena of silicate-magmas in general.

Neither "Soret's principle," nor any other form of the theory of diffusion, nor "connection currents," nor the magnetic attraction of the "liquid molecules," nor the different weight of the segregated solid constituents can afford an explanation of differentiation-phenomena of this kind. The geological conditions also seem to harmonize better with the view which connects the differentiation with the segregation and solidification of liquids insoluble in the remaining magma. These are separated out in consequence of the cooling of the magma, by which the conditions of solubility are changed, or of the escape of water or other mineralizers; the segregation takes place at first in the form of drops throughout the magma, which drops, on account of the surface-tension, have a decided tendency to coalesce and flow together into *schlieren* and larger concentrations. Which of the liquids is attracted to the side-walls and solidifies there, and which of them solidifies in the center, will in each case depend on the relative force of the adhesion to the side-walls.

Analogous Deposits.

Titaniferous iron-ore segregations in basic eruptive masses constitute a well-defined class, which has representatives in all parts of the world. Several of the different types found in Scandinavia occur in other countries also.

In the United States and Canada these ores have long been grouped together as a separate class, and a great many deposits of this kind have been described.

The magnetite-olivinite or Taberg type is analogous to the deposit at Iron-mine Hill in Cumberland, R. I., described by Wadsworth, though the latter is of smaller dimension. The

[23] Vogt, *Zeitschrift für praktische Geologie*, vol. ix., pp. 327 to 340 (1901).

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"gabbro titanic-iron-ores" of the Mesabi range in Minnesota, described by N. H. and H. V. Winchell, also seem to come very near to the type, though the concentration of the iron and the titanium has in these ores proceeded further. Among the segregations occurring in large masses in the various gabbro and labradoritic rocks of the eastern Adirondacks, the Taberg as well as the Ekersund type is represented.

Perfect analogies to the Ekersund type of ilmenite-segregations in labradoritic rock are offered by the Canadian ilmenite-deposits of Quebec and Ontario, which frequently contain from 30 to 40 per cent. of titanic acid and, in consequence thereof, a low percentage of iron. Some of these ores seem to consist of a mixture of ilmenite and titano-magnetite with a diminution of the percentage of titanic acid and an increase of that of iron. The inclosing rocks are labradorite and norite.

Deposits analogous to the magnetite-spinellites of the Routivare type also occur in the United States—namely, the chemically closely allied magnetite-spinellite deposits, accompanied by corundum, in the norites of the Cortland series, described by G. H. Williams.[24]

To the titanic ores of the nepheline-syenites correspond segregations of quite the same character in the rock-series of Magnet Cove.[25]

Utilization.

From the above it appears that the Scandinavian countries inclose very large supplies of iron-ores of this kind, and that, in reality, some of these deposits, such as Taberg, Routivare, Ekersund-Soggendal, etc., are among the largest iron-ore deposits in Sweden and Norway. Numerous attempts at exploiting them have been made in different parts of the two countries. Taberg has given rise to a local iron industry on a small scale carried on during two centuries. The ores of Ulfö have been used in several blast-furnaces in Norrland; from Ekersund-Soggendal during a succession of years ore was exported to

[24] "The Iron-Ore and Emery in the Cortlandt Norites," in Norites of the "Cortlandt Series," on the Hudson River near Peekskill, N. Y., *American Journal of Science*, Third Series, vol. xxxiii., p. 194 (1887).

[25] H. S. Washington. Igneous Complex of Magnet Cove, Arkansas, *Bulletin of the Geological Society of America*, vol. xi., pp. 389 to 416 (1900).

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England and smelted there; numerous minor mines scattered all over Sweden and Norway bear testimony to the attention which these deposits have attracted. However, all these attempts have been given up because of the unfitness of the ores for metallurgical purposes, which is also the cause why all or nearly all other titaniferous iron-ores all over the world lie unworked.

Group IV.—The Iron-Ores of the Metamorphosed Cambro-Silurian Schists.

These ores, which form a very well-defined geological class, are also territorially confined to a certain "ore-province." They occur exclusively within the area of more or less metamorphosed schists which forms the greater part of the mountain-districts of the Scandinavian peninsula north of the 65th degree of latitude. Through the abrasion of the Atlantic and the erosion the ore-bearing horizons have in places been laid bare; these ores are therefore almost exclusively confined to the Norwegian coast and the valleys penetrating into the country from the sea. The fjord-valleys deeply indenting the coast, as well as the nearest islands, are rich in deposits of this kind.

These ores have long been known, but, in spite of the favorable situation of several of the larger deposits on or near the Atlantic, they have been exploited, either not at all or to very small extent, by reason of their low percentage of iron. Only since the introduction of magnetic ore-concentration have attempts been made to utilize them for export on a larger scale.

The Geological Horizon.

The metamorphic schists in which these ores occur form a mighty complex of argillaceous and quartzite schists, gneisses, limestones and dolomites. Vogt has proposed to divide them into the Sulitelma schists, the younger gneiss group, and the mica-schist-marble group.

The whole of this series of strata has the character of a shallow-water formation, while the absence of conglomerates, the regular stratification, and the horizontal persistence of the rock-members, evidently prove that littoral formations do not enter into its composition. On the other hand, the abundantly thick limestones, and the more quartzose than argillaceous

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character of the schists, indicate that it is not a deep-water formation.

On the whole, one may distinguish in this rock-series an easterly, more phyllitic facies, which lies chiefly on the Swedish slope of the Scandinavian highlands, and a westerly facies, which, owing to the dynamo-metamorphic agencies accompany-

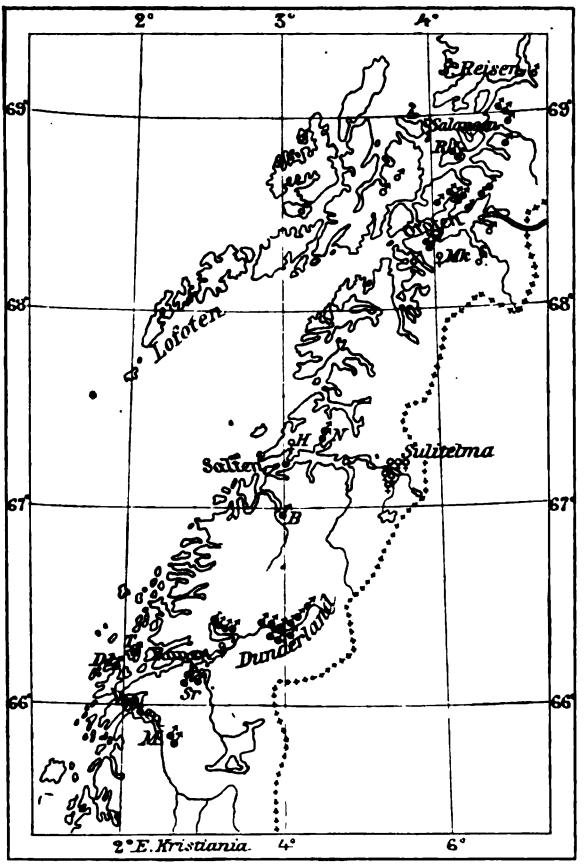

FIG. 19.—DISTRIBUTION OF ORE-DEPOSITS OF METAMORPHOSED CAMBRO-SILURIAN (Vogt).

ing the rock-plication, has received a more crystalline character. The degree of this crystalline character varies very rapidly, also, in a vertical direction. Thus the rocks of the so-called younger gneiss group show throughout a more crystalline development than the underlying schists of the mica-schist-marble group.

From fossils, chiefly found in the less metamorphosed schists

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on the Swedish side, all these formations have been shown to be of Paleozoic age. Also, in the vicinity of Sulitelma, on the very border between Sweden and Norway, fossils (encrinites) have been met with in a limestone, which occupies a lower level than the schists of the Sulitelma complex. These discoveries of fossils do not, however, afford an exact determination as to the age of the series.

The rock-plication within this area began in Silurian time, and continued long, ere it was completed during a post-Silurian, perhaps a post-Devonian, period. In connection with the rock-plication, there occurred great intrusions of igneous rocks in the form of laccolites, sheets and dikes. The intrusions are represented by basic as well as acid eruptive rocks, in composition ranging from peridotites, olivine-gabbros and other gabbro-rocks to light-colored granites rich in soda, forming together a rock-series of obvious "consanguinity."

Within this sedimentary complex the iron-ores occupy several horizons in the central and upper parts of the mica-schist-marble group. The ores seem exclusively to accompany the limestone and dolomite, an association which forms one of their most characteristic geological features. The known deposits of this class occur between Vefsen as the southernmost locality (about 65° 50′ N. Lat,) and Sydreisen in the province of Tromsö (69° 10′ N. Lat.), along a line of about 450 kilometers.

Dunderland.

The most important and best known of the deposits of this class are those of the Dunderland valley, which runs NE. from the interior end of the Ranen fjord. The Arctic circle crosses these deposits, which are distributed along a line of more than 40 kilometers.

The relations of the Dunderland field are somewhat complex, and have not, as yet, been sufficiently investigated. On the whole, the strata of the mica-schist-marble group are found in the bottom of the valley and the younger gneiss in the surrounding heights. But the details are much more intricate than Vogt's schematic representation[26] suggests. The inclination of

[26] Dunderlandsdalens Jernmalmsfelt, *Norges Geologiske Undersögelse*, No. 15 (1894), and Det Nordlige Norges Bergverksdrift, 1902, *Zeitschrift für praktische Geologie*, vol. xi., pp. 24 to 28, 59 to 65 (1903).

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the strata varies considerably; a change of the dip from nearly horizontal to highly inclined or vertical is observable within short distances. Owing to a strong plication the same ore-bearing layer will occur reduplicated, and the thickness of the limestone-beds is multiplied. Laccolites and sheets of light granite occur in many places between the stratified rocks without exercising any influence on the character of the ores. In general, this iron-formation, which may be regarded as a member of the stratified rocks, does not occur close to the limestone or dolomite, but is separated from it by a layer of barren schist up to a few meters in thickness.

The iron-formations may be followed for several kilometers without interruption; their thickness may amount to 100 m. or more, but sometimes diminishes to less than 10 m. Their contacts are sometimes sharply defined, but frequently the iron-formation passes gradually into the surrounding schists.

The iron-formation consists of the usual mineral constituents of the quartz-schists—*i. e.*, mainly of quartz and light mica, with which specular hematite (generally scaly) and magnetite are associated. Green pyroxenes and amphiboles are generally present, besides occasional epidote and garnet. The iron-percentage varies; the rock graduates on one side into the usual schist nearly free from iron, on the other into the richer ore-concentrations. Generally speaking, the iron-percentage varies between 15 and 40.

Within these primary ferriferous formations occur the richer iron-deposits which may be regarded as ores. In a few places these have attained an iron-percentage of about 55. This is, however, exceptional. Generally, the ore contains about 40 per cent. Since only about one-third of the iron occurs as magnetite, magnetic concentration is difficult.

Barren portions of the common quartzitic schist frequently occur inclosed in the ore. The ore contains the same secondary minerals as the iron-formation and in greater quantity. Only exceptionally is it mixed with lime.

The area of the outcrops of the workable deposits in the part of the field belonging to the English company, the Dunderland Iron Ore Co., is stated to be 577,000 sq. m. In this computation, however, part of the less rich iron-formation seems to have been reckoned in, and an iron-percentage of 32.5 some-

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times included. The average iron-percentage is said to be 38.6, and the quantity of ore that can be extracted by open cuts above the level of the valley is estimated at about 89,000,000 tons.

The greater part of these considerable ore-deposits was acquired in 1902 by the above-mentioned English company with a view to producing, by magnetic concentration, after the system of Edison, briquettes carrying 65 per cent. of iron from the 38 per cent. ore. At the same time the percentage of phosphorus is to be reduced from 0.2 to 0.3 to obtain an ore suited for the acid Bessemer process. In 1906 a railway from the mines to the harbor, concentration-works for nearly 500,000 tons of crude ore per annum, etc., were completed, but the export of the product had not commenced.

Naeverhaugen.

Among the iron-ore deposits of this class the Naeverhaugen field was the first that attracted attention. As early as about 1875 it was an object of several investigations both from a practical and a scientific point of view.

Naeverhaugen is of far smaller dimensions than Dunderland. However, its longitudinal extent is considerable, about 8 km. The general thickness of the ore-beds is only about 2 m.; exceptionally, it amounts to 10 m. or more. The whole ore-area has been estimated at 10,000 sq. meters.

The ore of the Naeverhaugen field consists mostly of specular hematite. As, consequently, a magnetic enrichment would be difficult to perform, attempts to concentrate in the wet way have been made, though without decided success. The field lies unworked.

Salangen.

The ores of this class occurring north of the fjord of Ofoten belong to a type different from the ores of Dunderland and Naeverhaugen; they consist chiefly of magnetites and, moreover, contain manganese. For the rest they show a similar mode of occurrence—viz., in regular strata and associated with limestones. An instance of this ore-type is offered by the field of Salangen, which has lately become the object of exploitation.

In the Salangen field the mica-schist-marble group is composed of quartzite-schists nearest to the fjord of Salangen,

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overlaid by mica-schists with garnet and amphibole, and with mighty interstratified limestone-beds. The ordinary light granite occurs as laccolites and sheets conformable to the stratification; in places one can observe how it is strongly impregnated with magnetite, which has been derived either from the schists at the time of the injection or by a secondary infiltration; a highly altered peridotite also occurs.

In several places considerable disturbances of the stratification occur in the form of inversions and plications. The ore-bearing formation almost always accompanies the limestone, and generally occurs in the schists quite near the limestone.

Contrary to the case at Dunderland, the ore in the field of Salangen consists mainly of magnetite. Experiments have proved that from three-fourths to nine-tenths of the iron-percentage can be magnetically concentrated. Minor beds of specular hematite in limestone occur. The iron-percentage generally varies in the ferriferous formation between 15 and 50. The richer parts, with 40 per cent. of iron or more, are of comparatively small extent, and the ore can be shipped and used only after magnetic concentration. The ore is characterized by a percentage of manganese generally varying between 4 and 6 per cent., but sometimes amounting to 14 per cent. The percentage of phosphorus is about 0.2, the ordinary percentage in these deposits.

In the ore occur the common gangue-minerals, amphibole, epidote, and garnet. The structure of the ore varies. Sometimes it is compact, coarse-grained and without parallel structure, as in the deposits of Storhaugen, probably derived by replacement of limestone. The ores which occur in the schists show the same structure as these.

The iron-ore field of Salangen was acquired at the end of 1906 by a German company, which intends to concentrate the ore and make it into briquettes after Gröndahl's method.

Other Deposits.

A great many ore-fields of this class occur within the area defined above. They differ, however, in minor points from those described. At Dolstadsåsen in Vefsen, the ore, which consists mainly of magnetite, occurs associated with amphi-

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bole, garnet, epidote, alternating with schists rich in amphibole between thick limestone-beds.

At Fuglestrand occurs a ferriferous formation in the form of schists rich in amphibole and epidote. Both magnetite and specular hematite occur. There are reasons for the assumption that the considerable development of iron-silicates characterizing the iron-formation of Fuglestrand depends on the abundant intrusion in this field of granite, which occurs as sheets parallel to the stratification as well as in the form of laccolites.

In the islands of Dönnesö and Tomö similar deposits have been the object of some exploratory work.

The Ofoten ore-deposits extend both on the northern and the southern side of the fjord. They belong to a basin-shaped complex of strata, cut transversely by the fjord. The tectonic conditions here are very clear. At the bottom of the mica-schist-marble group lies a thick bed of limestone, accompanied by the ferriferous formation, which on the south side of the fjord is strongly compressed, forming a basin with the axis of plication dipping towards NE. As youngest member occurs the younger gneiss. The Ofoten ores belong to the manganiferous type; the manganese is, however, very unequally distributed; it may amount to 10 per cent. The thickness of the ferriferous formations on the Ofoten exceeds only exceptionally 10 m.; in one place, however, a thickness of more than 50 m. is reported.

North of the fjord of Ofoten the same ferriferous formation is known in numerous places in the province of Tromsö. Besides the above-mentioned Salangen field, ores of this kind occur in the fjord districts of the Gratangen and the Lavangen, and in the islands of Rollö, Andorgö and Dyrö, where one and the same stratum forms flat basins, everywhere with iron-ores in the outcrop.

Summary.

The deposits of this group belong to a ferriferous formation of vast horizontal extent, occupying nearly the same geological horizon in the series as the mica-schist-marble member of this sedimentary series. The ferriferous formation occurs regularly associated with limestones, in most cases, however, in the schists underlying the limestone. The connection with the

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limestone is so strongly marked that in certain districts nearly every limestone-bed is accompanied by iron-ore; some observers have even been inclined to assume a connection between the thickness of the limestone-beds and the size of the iron-ore deposits.

Mineralogically, the ores are characterized as mixtures of magnetite and specular hematites; and, further, by the occurrence of iron-magnesia-lime-silicates of the amphibole, augite, epidote, and garnet groups. Quartz is always present in large quantity. Chemically, these ores are characterized by a high percentage of silicic acid, low percentages of CaO, MgO, and Al_2O_3, a medium percentage of phosphorus, and small amounts of sulphur and titanic acid. Whether these ores are primary sedimentary deposits or secondary concentrations of leaner iron-bearing formations is still an open question of the greatest practical importance.

These ores have, *e. g.*, in Dunderland, an extent of several kilometers in length and, at the same time, a considerable thickness. If, taking the syngenetic point of view, we regarded these ores as ordinary stratified formations, altered only by regional metamorphism, we would have to assume an extent of several kilometers in the direction of the dip. Such a conclusion might be highly misleading with regard to the ore-supply. As to the depth which the ores of this kind reach, there is as yet no practical experience, as neither exploratory work nor even any borings below the present or former ground-water-level have been performed.

A comparison with the Archæan ores of Group I. shows, it is true, some points of agreement. On the whole, the primary characters of chemical sediments are much more evident in this class of ores than in the Archæan ores. Above all, the transformations in the anamorphic zone are less marked; this is shown by the gangues being less developed and the alteration into magnetite less advanced. No analogies to the large magnetite-stocks among the Archæan ores are found here. Although the nearness to the magnesian limestone has offered plenty of material for the formation of lime-magnesian silicates of the pyroxene, amphibole, and garnet groups, these silicates never are formed in such abundance as to compare with the *skarn* gangue of the Archæan ores. The interchange of con-

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stituents between the schist, the iron-ore, and the limestone, lying close to one another, has been rather limited. Probably also the submersion in the anamorphic zone did not go so deep as to reach a temperature high enough for the formation of anhydrous silicates on a large scale. The depth and the temperature under which the crystallization of these ores occurred seem to have been better adapted to produce the slightly hydrated silicates of the epidote group.

The transformations in the catamorphic zone also are less marked; one finds no concentrations of so great richness, no *sköl*-formations, and no accumulations of ores in pitching troughs or on impervious basements. The ores are of more equal leanness, and in general the concentration does not exceed 40 per cent. of iron.

On the whole, one may consider these deposits as the roots or the deepest, comparatively unconcentrated parts of regional metamorphosed chemical depositions, laid open by the deeply penetrating fjords and valleys of the Norwegian coast; the upper, probably more concentrated and richer parts of the same deposits, having been destroyed by erosion.

Analogous Deposits.

The ores of this class are in many respects comparable to those of the lower Silurian formation in the eastern United States, especially to those occurring in a belt from Vermont to Alabama. These ores occur only where the lower Silurian limestone and the Hudson shales meet, as the Norwegian ores always occur at or near limestones, sometimes dolomitized. This is, indeed, one of their most important geological characters. The Norwegian rock-strata are, of course, more metamorphosed, the rocks consisting of crystalline schists and marble or magnesia-limestone, and the ores being specular hematite or magnetite, instead of red and brown hematite. The Clinton ores likewise show, with regard to geological conditions, great agreement with the Norwegian Paleozoic ores.

Group V.—Contact-Deposits in the Christiania Region.

These ores were considered by Keilhau, Daubrée and Kjerulf as genetically connected with the intrusion of granite in the

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same region; and this opinion was afterwards confirmed by Vogt, who executed a minute survey of the deposits.[27]

Classification of the Rocks.

According to Brögger, the igneous rocks of the Christiania region may be divided into seven groups of different age; the oldest three are more basic, and consist of (1) gabbro-diabases; (2) basic augite-, mica- and nepheline-syenites (laurvikite and laurdalite); and (3) quartz-bearing augite-syenite (akerite). Of later age is the following syenitic and granitic series, comprising (4) red quartz-syenites (nordmarkite); (5) soda-granites (grorudite); and (6) granitite. As the youngest members occur dike-forming diabase and diabase-porphyrite. The eruptives are probably of Devonian, surely of post-Silurian, age. They occupy an area about 250 km. long, and in some places more than 100 km. wide.

The intrusives are bordered partly by Archæan rocks, partly by the Silurian strata, and by porphyry-outflows. The contact-deposits are found in all these different pre-granitic rocks.

Most of the deposits are connected with the red quartz-syenite (nordmarkite), some of them with the soda-granite (grorudite) and the granitite.

The Ores.

The iron-ores are, as a rule, mixed with true contact-minerals, such as different species of the garnet and vesuvian groups, scapolite, wollastonite and others. This is especially the case with the deposits occurring in the limestone and marly slates. In the clay-slate, chiastolite is found as contact-mineral. The percentage of iron is through the gangue reduced to from 30 to 35, and only exceptionally exceeds 40. The ores are generally strongly pyritic, but low in phosphorus and titanium. Frequently they occur in the immediate vicinity of the intrusive rock, but they may also be found up to one km. from the contact. In a few cases the distance from the visible contact is more than one km., but the ores never are found outside of the metamorphic zone of the contact. The majority of the deposits occur in the Silurian strata, partly at the borders of the Silurian rocks, partly in big metamorphosed Silurian rock-fragments, completely surrounded by the igneous rock. The

[27] *Zeitschrift für praktische Geologie*, vol. ii., pp. 177, 464, (1894); vol. iii., p. 154 (1895).

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different Silurian horizons are equally impregnated with ores; and the clay-slates as frequently contain ore-deposits as do the limestones and marly slates. Also, the Archæan gneisses, and sometimes the porphyries, are ore-bearing.

The ore-deposits show generally a stratiform extension, and may be followed with varying thickness along the same bed for several hundred meters.

These deposits, though numerous, are quantitatively too insignificant to play any commercial rôle. In earlier times several hundred small ore-bodies were worked. The ore is mainly magnetite, but also specular hematite. Also, sulphides of iron and copper occur in so great amount that the ore may obtain the character of a copper-ore. Exceptionally, also, galena and blende have been found in such quantities as to be mined as ores. But all the deposits are small. From the seventeenth up to the latter part of the nineteenth century, they furnished a group of small blast-furnaces with iron-ores; and an insignificant copper- and lead-production was in early times based on these ores.

Analogous Deposits.

The ore-deposits of the Christiania territory are genetically most similar to the Pitkaranda deposit in Finland, and the known deposits of Schmiedeberg in Silesia and Berggiesshübel in Saxony.

There are a great number of deposits of this kind in the western United States, chiefly in Colorado and California, where they occur associated with the younger eruptives of the Rocky mountains and the Sierra Nevada.

GROUP VI.—LAKE- AND BOG-ORES.

These ores formed the raw-material for the oldest iron-industry in Scandinavia, long before the blast-furnace process was known. For this reason, Carl Linnæus called them *Tophus Tubalcaini*, after Tubal Cain, the first blacksmith (Gen. iv, 22). The lake-ores occur in most provinces of Sweden and in the southern part of Norway. But their abundant occurrence is confined to regions where the ground consists of moraine and glacial gravel and sand, especially the high plateau of Småland, the northern parts of Vermland, Vestmanland and Dalarne, and the greater portion of Norrland. They occur

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only sparingly in the regions covered by glacial and post-glacial marine-deposits, such as the lower coast-belt of southern Sweden, and the plains surrounding the great lakes of Vänern, Vettern, Hjelmaren and Mälaren. In short, the lake- and bog-ores are most frequent above the marine level of the glacial period. A certain connection with the distribution of the peat-mosses is indicated. On the other hand, the bog-ores are by no means more frequent in the districts rich in other iron-ore deposits. In some places, a connection with the greater pyrite-deposits may be suggested.

The bog-ores are recent formations, produced before our eyes. In lakes, where the ore has once been exploited, it grows and may be utilized again. In some lakes of Småland mining operations have been resumed at places exhausted 25 years before.

In the ore-bearing lakes, the iron is precipitated from dilute solutions chiefly along certain zones, parallel to the shores, at a depth of from 2 to 4 m., and the ores are thus distributed in belts on the bottom of the lake, to a thickness of at most 0.5 m. Lakes connected by a water-course frequently all contain bog-ores. In the upper lakes the ore is more fine-grained ("gunpowder-ore," "pearl-ore"), while in the lower lakes the ore has grown to coarser concretions ("money-ore," "cake-ore").

The purer lake-ores generally contain from 50 to 60 per cent. of Fe_2O_3 and from 10 to 15 per cent. of water. Silicic acid is frequently mechanically intermixed, reducing the iron-percentage. Sometimes the ores contain a considerable amount of manganese (up to 20 per cent.). The percentage of phosphorus as well as of sulphur is generally high.

Bog-ores are often formed in lakes as lake-ores, and later, by a natural draining, brought above the water-level. But frequently bog-ores occur in a manner indicating their formation in the ground close to the surface; in such cases they are often formed in connection with peat-mosses.

The utilization of the lake-ores has in later years much decreased, and is to-day without importance. From 1860 to 1875 the annual production in Sweden was about 10,000 tons. From 1900 to 1905, it was only about 1,000 tons, varying from 300 up to 1,500 tons, according to the severity of the winters. The whole product comes from Småland and is used for the fabrication of cast-iron.

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The Tar-Sands of the Athabasca River, Canada.

BY ROBERT BELL, M.D., LL.D., D.Sc., F.R.S., OTTAWA, CAN.

(Toronto Meeting, July, 1907.)

THE "Tar-Sands" is the name which has been given to the extensive horizontal deposit of fine Cretaceous sand, blackened by tarry petroleum, which forms the banks of the last or lowest 130 miles of the Athabasca river before it terminates in Athabasca lake. The Cretaceous strata, to which these sands belong, extend northwesterly from Dakota all the way to this lake and for 500 miles beyond it.

The Athabasca river forms the uppermost section of the Athabasca-Mackenzie, one of the great rivers of the world. It rises on the Pacific side of the Rocky mountains, westward of Edmonton, and flows north of east to a point called Athabasca Landing, 100 miles north of Edmonton, where it turns northward and runs about 280 miles before it falls into the west end of Athabasca lake. The section of this river-system which discharges the lake just named into Great Slave lake, is called Slave river, and below the latter lake it becomes the Mackenzie river.

Along the section of 280 miles of the Athabasca river above mentioned, the Cretaceous strata consist of bluish-gray and drab-gray marls, on top, from Athabasca Landing to Drowned rapid, 110 miles down stream; then heavily bedded gray sandstones, 225 ft. in greatest thickness, underlying these, form the banks for about 40 miles further. The strata appear to be horizontal, but they really dip very slightly to the southward. At Drowned rapid, we see the first of the tar-sands resting on almost horizontal beds of bluish-gray limestone of Devonian age. In some places the limestone is associated with shaly or marly bands. Notwithstanding the great hiatus between the two series, the horizontal tar-sands lie conformably upon the almost level upper beds of the limestone wherever the two formations are seen together, all along the river, throughout a

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distance of more than 100 miles. Slight local undulations occur in the limestone, but its general attitude is nearly horizontal. Its surface shows little erosion. We have here a rather puzzling phenomenon. How could an extensive horizontal surface of consolidated limestone retain a fresh and only locally eroded surface and then receive directly upon it a thick deposit of sand, also in horizontal beds and holding numerous seams of lignite? The erosions, occasionally seen, consist of shallow smooth-surfaced depressions scooped out of the uppermost beds.

Fort McMurray, 132 miles in a straight line upward from the mouth of the Athabasca, at The Forks or junction of the Clearwater river from the east, is a well-known land-mark, from which distances up or down the main river are measured. In descending the Athabasca, the tar-sands begin to appear in the banks at Drowned rapid, as already stated, about 30 miles above Fort McMurray, and the Devonian limestone is seen for the first time at Crooked rapid, at the level of the water, 20 miles above the same point. The limestone everywhere shows itself only at the edges of the river, and would be represented by a mere line on the geological plan. The fossils indicate that it should be correlated with the higher portion, and the Devonian system probably has a great thickness in this region. The Cretaceous rocks, all the way from Athabasca Landing to the mouth of the river, also appear to belong to the higher beds of this series. Although the tar-sands are seen in the banks of the river for a distance of about 110 miles, they are not known to extend far inland to the east; while on the west side, at a considerable distance from the river, they are probably overlaid by argillaceous bands the same as to the south. In the 30 miles above Fort McMurray, the tar-sands show a thickness of from 40 to 60 ft. in the banks, while for some miles below the fort they form the entire east bank, which rises from the edge of the water to a height, in some places, of about 200 ft. It is estimated that the greatest thickness of these sands is about 225 ft. From a point a few miles below Fort McMurray, the height of the east bank diminishes gradually in descending the river all the way to the delta, which is only slightly raised above the level of the water, and the blackened sand is visible nearly the whole distance, wherever the banks are not covered with trees. Figs. 1 and 2

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are photographic views of the east bank of the Athabasca river, 5 and 6 miles below Fort McMurray.

The tar in these sands has, no doubt, resulted from the evaporation and oxidation of the petroleum which passed upward through them during a considerable period of time. Along this part of the river the tarry sand either forms the surface of the country at the top of the banks or is overlaid only by till and drift sand and gravel, which to the southward contain no tar, although these deposits do hold rolled balls of mixed tar and stones and a few small patches of tar in some places further down the stream. It is therefore probable that, for a long time previous to the drift period, great quantities of petroleum escaped to the surface and were lost, or that much of the saturated sand was swept away before what now remains was covered by the drift-deposits. But to the south and west the tar-sands pass under the clayey marls, which would prevent the free escape of the oil to the surface.

The petroleum which saturated this Cretaceous sand-formation evidently came up from the Devonian limestones on which it rests, and probably from their deeper portions, as the limited exposures seen immediately below the tar-sands contain very little bituminous matter. In the cracks and joint-planes of these limestones, however, hardened tar or pitch may occasionally be seen, showing that petroleum has passed upwards through them at a remote period. The united thickness of the limestones and shales exposed in the upper beds along the river, from their first appearance, 20 miles above Fort McMurray, to the most northern outcrop, does not probably amount to 100 ft. The upper surface of the limestone, therefore, appears to slope northward with nearly the same grade as the river, which would average about 3 ft. to the mile or a total fall of 396 ft. As the Cretaceous rocks of the upper half of the distance to Athabasca Landing dip gently to the southward, a low E–W. anticlinal axis would appear to cross the river about Crooked rapid, or nearly midway between the lake and Athabasca Landing, always presuming that the Devonian limestone under the upper half of the distance continues to be conformable to the overlying sandstone.

In some places the Cretaceous sand-formation is not blackened by the tar, showing that the former escape of the pe-

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Dr. R. Bell, Photo.

FIG. 1.—TAR-SANDS, EAST BANK OF THE ATHABASCA RIVER, 5 MILES BELOW FORT MCMURRAY. (LOOKING SOUTH.)

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Dr. R. Bell, Photo.

FIG. 2.—TAR-SANDS, EAST BANK OF THE ATHABASCA RIVER, 6 MILES BELOW FORT MCMURRAY. (LOOKING NORTH.)

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troleum from the underlying limestones was not universal, so that possibly, at well-selected points in such unstained areas of the sand, there may be a better chance than elsewhere of finding liquid petroleum by boring to a considerable depth into the limestone.

The bitumen or maltha is, however, here present in far the greater portion of the Cretaceous sand-formation, which it supersaturates, giving it a coaly black color, and rendering invisible the numerous seams of lignite which it contains. Under the hot summer sun, the tar-sand melts at the surface and flows down the steep naked banks, rolling over and incorporating dry sand and stones, which have fallen from the drift above it; and breaking away in balls and rolls, it accumulates along the foot of the bank. The sand is fine-grained, and at cool temperatures the mass may be chopped out in lumps without any visible effect on the edge of the axe. When piled upon a wood fire, the lumps burn like cannel coal for some time and then collapse into sand.

In going down the Athabasca river, the tar-bearing sands are first seen, as already mentioned, at Drowned rapid, 30 miles above Fort McMurray. From this rapid it is exposed, with some intervals of concealment, all along the river to a point 9 miles below the junction of the Calumet from the west, a distance of 90 miles, but it probably continues under the surface-deposits nearly to the head of the delta of the Athabasca, or a total distance northward of at least 100 miles. The breadth of the country westward of the river which is underlaid by the tar-sands is uncertain, but if it should not exceed 30 miles, a low estimate, this tar-soaked formation would have an area of 3,000 sq. miles on the west side of the Athabasca. It is seen also almost continuously along the east side of the river from Drowned rapid nearly as far as on the west side, and no doubt also continues under the low country approaching the delta. It does not, however, appear to extend as far east as it does west from the river. In ascending the Clearwater river, which joins the Athabasca from the east at Fort McMurray, large masses of the thickened sandy tar, similar to those which roll down the steep banks of the Athabasca, were found in the bed of the stream at 11, 13 and 17 miles from its mouth, and these probably indicate the existence of the

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tar-sands concealed under the moss and trees which cover the banks.

At Drowned rapid the tar-bearing formation is only 10 ft. in thickness, but it increases steadily until it has attained 200 ft. a few miles below Fort McMurray, and its maximum is apparently about 225 ft., although it may be greater under the low ground of the northern part of its area. If its average depth should be only about 100 ft., the total quantity would still be enormous.

The tar-sands and liquid tar of the Athabasca have been known to the fur-traders of the Mackenzie River district from the earliest times. They were mentioned by the first travelers that wrote about the country, particularly by Sir John Richardson, who described this part of the Athabasca in 1823. They were noticed along with other occurrences of petroleum in the Northwest Territories in a paper which I published in 1881,[1] On the Occurrence of Petroleum in the Northwest Territories with Notes on New Localities. The geological relations of these deposits, their distribution, volume, origin of the petroleum, etc., are fully described in my accounts of this region.[2] Additional information is given in the report of Mr. R. G. McConnell, ten years later,[3] on A Portion of the District of Athabasca. Owing to the absence of railways in the district, but little effort has yet been made to bore for liquid petroleum under the surface.

In 1897 and 1898 a bore-hole was sunk, at the expense of the Geological Survey,[4] on the west side of the river at Pelican rapid, 80 miles in a straight line above Fort McMurray. At a depth of 820 ft. a great flow of gas at high pressure was encountered, and this effectually prevented further boring. The gas has continued to escape with a roaring noise, through a 4-in. pipe, during the nine years which have elapsed since it was first struck. For some years, the noise was so great that it could be heard at a distance of two miles or more, especially

[1] *Proceedings of the Canadian Institute*, Series III., vol. i., pp. 225–230 (1879–83).

[2] *Report of Progress of the Geological Survey of Canada*, 1882–83–84, Part C C; also *Summary Report for* 1889, pp. 103 to 110.

[3] *Annual Report of the Geological Survey of Canada*, vol. v., part I., report D (1892).

[4] *Summary Report of the Geological Survey of Canada for* 1897, pp. 18 to 25; *Summary Report*, 1898, pp. 28, 32 to 34.

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in the winter. After nine years the pressure appears to be lessening, although this may be due to some obstructions in or below the casing. When the gas was originally encountered a little thick, dark petroleum was said to have been blown out of the bore-hole.

The black and viscid tar which saturates the Cretaceous sands so abundantly in the large area above described, has no doubt resulted, as above stated, from the partial evaporation and the oxidation of a more fluid form of petroleum that entered the formation from below. Its original source has probably been in the deeper strata of the Corniferous (Devonian) limestone underlying the sands. It is possible that it came from a formation or formations below the Corniferous; but the tar is so generally diffused in the sand that it is more probable it came from the formation immediately below it, because the deeper the oil-producing strata the fewer points of escape it would have by the time it reached the Cretaceous sand. The numerous exposures of Corniferous limestone seen under the tar-sands showed little evidence of containing bitumen, except as black incrustations in joint-planes, cracks and vugs, which were observed in some places.

The geographical area of the tar-sands, the thickness of the deposit, as well as the large proportion of the contained tar, all indicate an enormous quantity at the original source of supply. The saturation of the sand is not uniform throughout. On the contrary, in a few localities no oil at all appears to be present, while in others the sand shows different shades of brown and black, according to the amount of petroleum it has received.

In any petroleum-bearing region the oil is believed to be held under impervious strata in low anticlinals or domes until the accumulation is punctured by an artificial boring. In the region in question, several local anticlinals were observed in the beds of Devonian limestone and marl immediately under the tar-sands. Any accumulations of petroleum which may still exist at greater or less depths in the limestone, or in formations underlying it, probably remain in such anticlinals and domes, the arrangement of which may be quite independent of that of the overlying unconformable tar-sands. The area of the deep-seated petroleum-bearing rocks being large and the conditions varied, it may be reasonably supposed that all the oil has not yet escaped,

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but that considerable quantities may still be imprisoned at various depths in some parts of the field.

The complete or partial absence of tar in the sand at any particular place does not necessarily indicate that a certain share of the petroleum, which had elsewhere escaped upwards into the sand, still remains below such a locality. Its absence may be due to its having been carried past by a slight ascending grade along the axis of an anticlinal, or owing to the spot being situated over a basin or a synclinal form in the strata, or to some other cause not readily explainable.

If the main or general anticlinal of the whole district, as indicated above, runs approximately E–W., which appears probable, any subordinate anticlinals may be expected to have a somewhat similar course. The evidences of very profuse outpouring of petroleum towards the surface in former times may not be the best indication of its present existence at considerable depths, since it may have escaped so freely as to have left but little behind. A careful geological survey of a wide enough area should be made, and the exact direction and amount of the dip of the Devonian rocks at numerous spots should be marked with precision at each particular locality upon a large-scale map, which should also show the relative proportions of the tar in the overlying sand-formation, and all other facts that might aid in determining the most promising sites at which to locate trial borings. This being a new and practically untried oil-territory, it may be found that some peculiar conditions exist, a knowledge of which is necessary to success; so that, after having taken all such precautions as those above mentioned, numerous experiments may be necessary at first to discover the mode of occurrence of the oil and the best method of reaching it by boring.

But even if it should be proved that no liquid petroleum was to be found by sinking wells in this region, the bitumen, tar, or maltha of the sand itself exists in such enormous quantities that it will, no doubt, be utilized in the near future. On the assumption that the saturation of the fine sand by the tar extends from the bottom to the top of the formation over the whole area in which this sand forms the surface, and after making allowance for the unimpregnated spots, the quantity of this material in the Athabasca district would amount to

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from 50 to 100 cubic miles, or many billions of tons, and it would be practically inexhaustible. Professor H. V. Winchell, in discussing this point with me after the reading of this paper, thought it was quite possible that the tar was more concentrated towards the surface of the country and the face of the banks than throughout the mass; but the saturation appears to have taken place before the present banks were formed. I am not aware of any other place in the world where there is such an immense surface-showing from the uprising of petroleum. Owing to the evaporation of the lighter portions of the original oil, its composition cannot now be known, and therefore the total quantity of such petroleum, at present represented by the bitumen in the tar-sands, cannot be estimated. The portions of the formation which have been removed by denudation would greatly increase the amount.

Although little or no bitumen is present at a few localities, and in others there has been a concentration by the draining-out of the tar from the mass, still the coal-black sands which form the great bulk of the formation appear to have a fairly uniform composition. An average fresh sample of this material, collected by me, having a specific gravity of 2.040, was analyzed by Dr. G. C. Hoffmann, late chemist to the Geological Survey, who found it to contain 12.42 of bitumen, 5.85 of mechanically included water, and 81.73 per cent. of very fine siliceous sand. At a temperature of 50° F. it was quite firm, barely, if at all, yielding to pressure, and did not soil the hand; at 70° F. it gave somewhat to the touch, and was slightly sticky; at 100° F. it became quite soft and eminently soiled the fingers.[5] Lieut. Cochrane, chemical instructor at the Royal Military College, Kingston, and Mr. Isaac Waterman, petroleum refiner of London, Ontario, each found about 15 per cent. of bitumen in average samples of the tar-sand which were submitted to them.

The percentage of bitumen which may be found in samples of the tar-sands depends largely on their selection. The melting and flowing-out of the supersaturating tar during the summer produces a much richer variety than the average. Dr. Hoffmann found that a specimen of this kind had a specific gravity of only 1.023, or half that of the sample he analyzed,

[5] *Report of Progress of the Geological Survey of Canada*, 1880–81–82, part H, pp. 3 to 5.

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which was no doubt due to the much smaller percentage of sand which it contained.

The heat required for the treatment of the tar-sand may be obtained from this material itself by burning it in lumps on a grate made of corrugated iron, thickly pierced with holes, similar to the grates used for burning very damp sawdust at many Canadian mills, with an additional contrivance for removing the sand, which may be used for making the finest glass. The oils may be extracted by one of the two or three processes which seem to be available, after experiments have determined which of them is the best. The late Dr. T. Sterry Hunt suggested that the lighter oils first obtained by distilling the tar-sand may be used to dissolve out the oil from a fresh portion of the raw material, while the subsequent heavier portion that came over constitutes a valuable lubricating oil. As the fuel and material treated cost nothing but the handling, this may prove a cheap method of extraction.

Dr. Hoffmann found that 70 per cent. of the bitumen contained in the sand can be extracted in a fluid state by boiling it in water. The extracted oil rises rapidly to the surface and may be drawn off. By either of these methods, the great bulk, say 90 per cent., of the crude material may be rejected, thus reducing very much the labor required for distilling and refining the valuable portion, which will then be confined to the bitumen that separates in a fluid state. The pools of pure tar and the pitchy deposits, formed by the natural process of concentration already described, may be found to occur in sufficient quantities to be utilized in the manufacture of refined oils.

Different experiments made with the tar-sands show that, while they yield some good illuminating fluid, their principal value consists in the large proportion of fine lubricating oil which they afford. This oil was found to remain liquid at the cold winter temperatures of the Canadian prairie provinces, and therefore it is very suitable for car-wheels and machinery working in the open air in these or other cold regions.

The high banks of the Athabasca river and its branches in the tar-sand area offer great facilities for excavating this material; and as it occurs in such unlimited quantities, and can be taken out in lumps suitable for burning at a trifling cost, there

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will be no restriction on its free consumption, one portion of the raw material being used to produce oil from another. When this part of the country becomes inhabited, and towns and villages spring up in various directions, the tar-sands may be used for roofing and paving, and also for the manufacture of illuminating gas, as well as oil.

From what has been stated above, it is reasonable to expect that this immense deposit of tar-sand in the Athabasca district may be destined to prove, at some time, of great economic importance. The material, just as it occurs, with little or no artificial treatment, is ready to be utilized as asphalt for paving, roofing, electric insulation and other purposes. The natural intimate combination of the fine sand and bitumen imparts to it a quality which no artificial mixture of similar components could be expected to possess. Some cheap means may be found for its transportation to towns and cities where it will meet with a ready market.

Dr. Hoffmann found that the sand, which forms about 80 per cent. of the deposit, consists of grains of pure vitreous quartz, suitable for the manufacture of the finest white glass, so that we have here both the fuel and the silica for glass-making. The Geological Survey has proved that common salt exists abundantly in a dry state and also as brine in many places all along Salt river, a branch of Slave river, which is the next section of the Mackenzie below the Athabasca; and also at a place called La Saline, on the latter, where the brine flows over a wide, smooth surface of the tar-sand, leaving conspicuous deposits of salt in the neighborhood. No doubt great quantities of brine could be obtained by boring into the Devonian strata, from which the brine proceeds, so that an abundant supply is available for conversion into sodium carbonate, to use with this excellent form of white silica for the manufacture of glass, for which the tar-sands would furnish cheap fuel at any locality where the works might be located. It is probable that natural gas could also be found by boring.

In regard to a market for the oils which may be produced from the tar-sands or by sinking wells, it may not be a long time before this region is reached by railway from the south. There is already a considerable demand for illuminating and lubricating oils in the prairie provinces and British Columbia.

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If the output should become very large, the refined oil could be piped to Fort Smith on Slave river, and thence sent on tank steamers, by way of Mackenzie river, to the northern ocean, which is open from the mouth of this great stream to the Pacific. Or it might be carried by steamers to the waters east of Athabasca lake, and thence piped to Churchill harbor on Hudson bay.

Destruction of the Salt-Works in the Colorado Desert by the Salton Sea.

BY WM. P. BLAKE, TUCSON, ARIZ.

(Toronto Meeting, July, 1907.)

THE salt-beds at Salton, on the line of the Southern Pacific railway, in San Diego county, California, have been successfully worked for many years by the corporation known as the East Liverpool Salt Co., and the salt has been largely used upon the Pacific coast generally and in the interior. But the industry has lately been destroyed by the overflow from the Colorado river. The salt is dissolved, and the waves of the Salton sea now roll 50 ft. above the site of this once thriving and growing business.

The salt-bed was evidently the residual accumulation from the waters of an ancient lake, which I have named Cahuilla, from the name by which a large part of the valley—the home of the Cahuilla tribe of Indians—is known. The deposit covered a comparatively small area in the lowest part of the valley, 280 ft. or more below the sea-level. This depressed area is now generally known as the Colorado desert, but has a soil of wonderful fertility wherever water can be had on its surface.

The deserted beaches, and the water-lines, only a few feet above tide-level, upon the rocks of the bordering mountains, as well as a great abundance of shells in the soil, all bear decisive testimony to the former occupation of the valley by water.

The depression is topographically the northern end of the Gulf of California, from which it is now separated by the delta of the Colorado river, extending practically from the mouth of the Gila river across the valley to the coast mountains on the

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west side. We may conclude that the imprisoned salt water was gradually displaced and freshened by the inflow and outflow of the river, so that the lake was changed to fresh or brackish water, with enough salt in it to give a bed at the bottom upon the final evaporation of the lake-water after the Colorado had shut itself off.

It is also possible that the salt-bed owed its origin to springs of salt water, which are known to exist, and are by some supposed to indicate a connection with the water of the Gulf.

A deep valley lying west of the Cocopah mountains is occupied by a lake of salt water, known as Laguna Salada. It appears to receive occasionally accessions of sea-water and also of fresh water, by the way of Hardy's branch of the Colorado delta.

It is estimated by Mr. Arthur P. Davis, Assistant Chief Engineer of the U. S. Reclamation Service, that, if the inflow of the Colorado to the Salton is prevented, the sea will dry up by evaporation in about 10 or 12 years. Under such conditions we may expect the restoration of the salt-bed and a possible renewal of the industry.

Ore-Deposits of the Eastern Gold-Belt of North Carolina.

BY W. O. CROSBY, BOSTON, MASS.

(Toronto Meeting, July, 1907.)

INTRODUCTION.

THE crystalline belt of the Atlantic Seaboard, south of New York, attains its maximum breadth of 220 miles on the northern border of North Carolina; and in this State it is most widely characterized by the occurrence of gold in workable deposits. Half a dozen auriferous belts have been recognized by Nitze and Hanna.[1] These are, in the main, coincident in position and trend with the NE.–SW. zones of metamorphic slates and schists, which, more than any other feature, give character to the geological map of North Carolina. The central gold-belts, accompanying the main zone of slates and as-

[1] *North Carolina Geological Survey, Bulletin* No. 3, p. 15 (1896).

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sociated igneous rocks, are much the more important and persistent, and may be regarded as continuous northward across Virginia and southward into South Carolina. The eastern belt, on the other hand, is the shortest and, economically, one of the least important. Its extreme length does not exceed 25 miles, the narrow band of slaty rocks to which it belongs passing, both NE. and SW., beneath the coastal-plain formations. But, although limited in area and production, this belt is, perhaps, comparable with the central belts or any part of the seaboard crystallines in the variety of conditions under which the gold occurs. It lies chiefly in the NE. part of Franklin county and the NW. part of Nash county, on the watershed of Fishing creek and other tributaries of Tar river. The elevation is between 400 and 500 ft.; and the Tertiary peneplain has here an exceptionally perfect development, the reliefs separating the slightly incised stream-courses consisting of broad, flat-topped peneplain remnants, bounded by gentle lateral slopes.

General Geology.

The eastern gold-belt is separated from the central gold-belts by the granitic rocks of the Raleigh, Louisburg and Warrenton area; and these plutonic rocks are clearly younger than, and intrusive in, the associated metamorphic sediments. In traveling east from Louisburg or SE. from Warrenton to the mining district, it is evident that we pass from a complex of granitic and gneissic rocks and various schists to an area of prevailing quartzites and siliceous slates, which are, however, freely intersected by the plutonics. Presumably, the sedimentary rocks are wholly of early Paleozoic age; and certainly their relations to the igneous rocks, so far as observed, are essentially identical throughout the district. The general line of strike of the sedimentary formations, as previously indicated, is NE.–SW.; and the prevailing dip is SE. at high angles. No effusive rocks, either acid or basic, have been observed in this district.

Almost universally, the rocks, of whatever kind, including even the hardest and most siliceous quartzite, are deeply rotted to normal saprolites; and both lithologic and structural features are thereby greatly obscured. Furthermore, the upland or peneplain surface is, very generally, overspread by a mantle

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of fine quartz sand. In part this sand appears to be simply the saprolite phase of the quartzite; but elsewhere it is horizontally stratified and includes, at least basally, layers of quartz gravel; and it seems then to require correlation with the Columbian formation.

Genetic and Structural Relations of the Gold-Bearing Formations.

Sedimentary and Plutonic Rocks.—First, in order of age, come the sedimentary rocks, consisting chiefly of normal quartzite, grading through siliceous slates into metamorphic clay slates or phyllites. Intersecting these terranes in a highly irregular manner and often transversely is the complex of plutonic rocks, among which diorite, normal granite and pegmatite are especially prominent.

The diorite is believed to be here, as in so many other plutonic regions, an earlier, peripheral, basic phase of the same body of magma which has yielded the granite; and it is clearly cut by dike-like masses of granite. Both the diorite and the granite exhibit marked variations in texture and composition. They are frequently gneissic; and a sheared and chloritized part of the diorite is the chloritic schist of other writers.

The pegmatite, on the other hand, is clearly a later phase of the granitic magma, and forms irregular dikes in all the other rocks of the complex, both igneous and sedimentary. So far as noted, it is of highly acid composition, consisting chiefly of megacrystalline quartz, acid feldspars, and muscovite.

Veins.—In part, the pegmatite is highly quartzose, and shows an unmistakable tendency to grade into quartz-veins, which are the feature of the complex of most particular interest, so far as the occurrence of gold is concerned. This tendency, and the occasional occurrence of coarsely crystalline muscovite, identical with that of the pegmatite, in otherwise normal auriferous quartz-veins, leaves no room to doubt the genetic relation of the pegmatite-dikes to some, at least, of the quartz-veins, or that the gradation is a fact.

Large areas of the complex of sedimentary and plutonic rocks are traversed by an irregular network of quartz-veins, varying in width from a line to 50 ft. or more, but mainly rather narrow, and, individually, highly variable in size and

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trend. They are, without exception, massive and glassy; and bands, combs, vugs, or other indications of crustification or of endogenous origin are conspicuous by their absence, as are also the selvage-phenomena of normal fissure-veins. The veins have not, as a rule, been worked below the oxidized zone, although this is shallow; but indications of sulphides below the water-level are very meager and often wholly wanting. Aside from the oxidation of the scanty sulphides, the quartz-veins have, in general, survived the rotting of the inclosing formations, and are now traceable, *in situ* and essentially intact, through the residuary detritus or saprolite.

In the central gold-belts of North Carolina, I long since recognized two essentially distinct types of auriferous veins: (1) Massive quartz-veins of varying and often considerable size; devoid of definite selvage, combs and vugs; relatively, and often absolutely, barren of sulphides; and the average gold-tenor low to very low, but highly irregular or pockety, the gold being free, mainly coarse and nuggety, and thus especially favorable for the enrichment of placers. (2) Relatively narrow quartz-sulphide veins, with the normal structural features of true fissure-veins; and relatively high and even gold-values, the gold being rarely coarse and only to a limited extent free.

The veins of the first class are believed to be, in general, the older; in part, approximately contemporaneous with the granitic rocks; and hence to have been formed under essentially plutonic conditions by so-called magmatic waters, and to admit of correlation, to some extent, as the end-term of the pegmatites; while the veins of the second class are the product of the more normal circulation of the ground-water at less than plutonic depths, and represent the leaching of the upper and less metamorphic zones of the earth's crust by waters largely or chiefly of meteoric origin.

The veins of the second type form the basis, now and then, of continuously profitable mining operations; but those of the first type are rarely payable except locally and transiently, where a pocket or bonanza is encountered. These are the "specimen" veins, which are most profitable to man where they have been extensively mined by the slow processes of erosion and the product concentrated in placer-deposits. Again,

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having been formed under approximately plutonic conditions, and not being in any large degree dependent upon sulphides for their values, the veins of the first type are not, to any important extent, subject to secondary enrichment by downward leaching. In other words, their marked variations of value are primary and do not, as in the sulphide veins, admit of close correlation with depth, both the coarseness of the gold and its uneven distribution being consonant with a quasi-plutonic origin, which has allowed the principle of segregation to accomplish its most perfect work, bunching the gold in pockets within the veins, and in nuggets within the pockets.

Obviously, the veins of the eastern gold-belt belong to the first or non-sulphide type; and in this fact we find the key to the history of gold-mining in this district. Also, it may be noted here that the best values are found, as a rule, in the narrower veins, the wider ones being of little or no economic interest.

Impregnations.—The immediate country-rock, be it granite, diorite, or quartzite, of the quartz-veins and veinlets is often slightly impregnated with gold; but it is doubtful if the degree of impregnation is ever sufficient to make the undecomposed hard rock a practical ore. With the saprolite, however, the case is somewhat different, since it is easily mined and requires no crushing. The mineralized or impregnated granite saprolite, known locally as the "white belt," usually carries the best values. We are considering here a type of deposit—metasomatic impregnation and replacement—which elsewhere in the southern Appalachian region has proved of exceptional importance from a mining point of view. But in this eastern gold-belt it is, even when reduced to the condition of saprolite, of little economic interest, the values being, in general, still too low to justify metallurgical treatment, and the proportion of coarse gold too low to permit a large saving by placer methods. In comparison with the veins, the impregnations are noteworthy for the finely divided condition of the main part of the gold, and for its relatively even distribution. As in the veins, however, sulphides are meager or wholly wanting. In brief, the impregnations partake of the character of the veins which they border, save that the gold is prevailingly fine instead of coarse, and hence unfavorable for the enrichment of placers.

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Placers.—Long continued and extensive erosion, resulting in the peneplanation of this region, set free a notable amount of the coarse gold of the quartz-veins and effected its concentration in residuary placers adjacent to the outcrops of the veins from which it was derived. Subsequently, during the Columbian submergence, this residuary gold was still further concentrated, as a marine placer, in the basal layers of the Columbian gravels. Finally, during the erosion of the gravels and the trenching of the underlying peneplain by the modern valleys, the gold again became a residuary deposit; but in large part, also, it was carried down the slopes and experienced a further concentration in the gravels of the stream-beds. Thus, for the surficial, as for the bed-rock formations, we may recognize two principal types of auriferous deposits: (1) residuary and marine placers, representing a primary, vertical concentration in a plane, during the development of the peneplain; and (2) stream placers, representing a secondary, horizontal concentration in a line, during the trenching of the peneplain.

The bed-rock deposits—veins and impregnations—are, as we have seen, of little economic interest, except locally for the saprolite phase; but where nature has mined these formations by cubic miles and concentrated the gold in the ratio of thousands into one the results may be very interesting, although the deposits are quickly exhausted and do not afford a basis for long-lived mining.

Mining Notes.

My observations in the eastern gold-belt were chiefly confined to three properties: but these appear to illustrate satisfactorily the normal range of conditions for the district.

Alston Mine.—This property is a part of the plantation of Edward Alston, in Warren county, about 16 miles SE. of Warrenton, and on the NW. margin of the gold-belt. Gold was first discovered here in 1847 through the finding of a nugget in the road by one of Mr. Alston's slaves. The situation is a low knoll of saprolite in which may be readily recognized, in order of age, quartzite (sometimes micaceous and garnetiferous), diorite (fine, dark and chloritic), granite (fine, white and micaceous), dikes of pegmatite, and a plexus of quartz-veins. The superficial, residuary deposit or placer has been worked over an area of an acre or more, yielding a considerable amount

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of coarse gold. Several of the quartz-veins, prospected to the water-level (from 25 to 30 ft.), have also yielded good values; but they are much too small to admit of profitable mining. The fine, impregnating gold is chiefly confined to the vicinity of the quartz-veinlets in a band (dike) of granite saprolite from 20 to 30 ft. wide, forming the "white belt" of this property. In the pan, this impregnating-gold appears as numerous, very minute colors of remarkably uniform size, a condition highly characteristic of this type of deposit and making impossible any large recovery by placer methods.

Sturgess (Portis) Mine.—This mine, the best known and most productive of the eastern gold-belt, is situated near Ransom's Bridge in the NE. corner of Franklin county, at an elevation of about 100 ft. above Shocco creek. The extent of the workings and the variety of deposits which they exhibit make this property exceptionally favorable for the study of the geologic relations of gold in this district.

The predominant bed-rock formation is the diorite, which is, at most points, thoroughly rotted, yielding, superficially, a deep red soil. Cutting through the diorite saprolite are occasional dikes of fine-grained granite saprolite (white belts). One of these is only from 4 to 5 ft. wide, while another is more than 70 ft.

Both the diorite and the granite are traversed by a network of very irregular quartz-veins, of all sizes up to 2 ft. thick. They are said to be generally auriferous, with pockets running thousands of dollars to the ton, the small veins being relatively the richest. The granite saprolite or white belt also carries appreciable, and, in part, workable values, although the minute subdivision of much of the gold prevents a high recovery.

As usual in the eastern gold-belt, the surficial deposits have been most productive. These include: 1, a considerable area of Columbian gravel, the basal layer of which, resting unconformably upon the diorite and granite saprolites, contains workable values; 2, a still larger area of residuary placer, where the Columbian gravel has been removed by erosion and the gold left directly upon the surface of the underlying saprolite; and 3, the stream gravels in the bed of a minor water-course or "branch," the valley of which trenches the saprolite.

Thus we may recognize here, in obvious genetic relation, five

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available sources of gold; two original or bed-rock sources—the quartz-veins and the accompanying impregnations of the granite saprolite or white belt; and three secondary or placer sources—marine, residuary and stream.

The Columbian gravel is still largely intact; but the residuary and stream deposits have been worked to the verge of exhaustion; and to these may be credited a large part of the total product. In part, also, the quartz-veins and white belts have been worked to depths of from 20 to 30 ft., the saprolite being washed, and the quartz trammed to a 20-stamp mill and crushed. The chief handicap to the operations has always been the lack of water, which can be had in quantity or under pressure only by pumping. Were a gravity supply available, hydraulicking of the entire property down to the "branch," followed by the milling of the quartz thus mined, and the subsequent cleaning up of the "branch" with a dredge, would, doubtless, yield interesting results.

North Carolina Placer Mine.—The property of the North Carolina Dredging Co. is on the same "branch" which traverses the Sturgess mine, and immediately below the latter. The valley-floor is here several hundred feet wide, with a very low gradient; and the conditions are in every way favorable to the operation of a dredge. A low dam has sufficed to create the necessary pond for a Robinson chain-bucket dredge with a daily capacity of 1,000 cu. yd., working 14 ft. deep. The "pay" streak is a layer of clean gravel, from 6 in. to 3 ft. thick, on bed-rock, with a layer of tough clay over it. Some gold is found in the silty overburden; but it is mainly in the gravel; and the values are fairly even across the valley, although greatest near the middle. The gold is mainly fine to very fine, or such as might have been transported by this feeble stream during a recent period of elevation, when it was flowing on bed-rock.

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Deutschman's Cave, Near Glacier, B. C., Canada.

BY W. S. AYRES, BANFF, ALBERTA.

(New York Meeting, April, 1907.)

I. Introduction.

This cavern was discovered Oct. 22, 1904, by Mr. Charles H. Deutschman, in company with whom I made, May 29 to June 3, 1905, at the request of Mr. Howard Douglas, Superintendent of the Canadian National Parks, the first exploration of it for the Canadian government. The results were stated in a paper, orally presented, with lantern-views from my photographs, at the British Columbia Meeting of the Institute, Victoria, B. C., July 5, 1905.* The present paper embodies also the results of a second examination, made Oct. 25 to 29, 1905.

The accompanying map, Fig. 1, will serve as a guide to the following description.

II. Location.

The cavern is situated at snow-line, at the head-waters of Cougar creek, on the west slopes of the Selkirks, about 2 miles north from Ross Peak water-tank, and 5.5 miles west from Glacier station, on the main line of the Canadian Pacific Railway. It was then reached by an arduous climb from the water-tank, up the ravine of Cougar creek, for 8,000 ft. (2,000 ft. of altitude) over rock- and snow-slides, and through a thick tangle of alders. But an easy and very picturesque trail has, during the past summer, been opened by the government from Glacier station, which is also a hotel of the railway company.

Figs. 2, 3 and 4 represent views, not heretofore published, of the grand scenery of this region, so rapidly becoming the object of attraction to thousands of tourists from all parts of the world. These views, as well as those shown in Figs. 5, 6, 7, 8

* Withheld from publication, to await the later exploration here mentioned. See *Trans.*, xxxvi., liv. (1906). The illustrations accompanying this paper have been selected as typical from a much larger number of photographs taken above and under ground by the author.—R. W. R.

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and 9, were taken from different points in the neighborhood of the cave, or on the way to it. Many other beautiful and sublime pictures of glaciers, twin lakes, forests and cascades could be added from the same region.

III. The Rocks and Their Formation.

The cave occurs in hard crystalline limestone, dipping about 30° to the east, and, according to the Canadian Geological Survey, belonging probably to the Devonian age. The beds are very thick, and are made up of alternate bands of white, mottled, and gray marble. Some of the bands are so highly impregnated with fine grains of sharp sand that excellent whetstones can be made from them. The limestone rocks have not been completely changed into marble at all points, as will be noted under the next head, yet the change has been sufficient to obliterate all fossils.

The cave has undoubtedly been formed almost entirely by water-erosion, no part of it showing any extensive evidence of a slow dissolving of the rocks. Cougar creek, which formed it, is entirely made up of glacial- and snow-water, and during the spring and early summer is a mountain torrent. The fine grains of sharp sand loosened from the limestone rock and caught in the swift current of the small stream, that at first found its way through a shrinkage-crack or a fault in some particular bed of limestone, have undoubtedly given the water an uncommon erosive power, and enabled it to carve out a mammoth subterranean water-way in solid marble. The almost total absence of stalactites and stalagmites, such as are usually found in caves, and the presence of curiously carved marble walls, of varied fantastic shapes and somber coloring, are unusual features.

IV. Cougar Creek and the Cave.

Some distance above the cave, the creek passes for 350 ft. under a natural bridge, called by us "Gopher" bridge, and this passage is of itself a picturesque cavern. Besides the characteristic water-carved walls of white and gray marble, everywhere exposed, an additional feature is presented by the circumstance that, in many places, the change of the limestone into marble has not been complete, and the incompletely altered parts of the rock stand out in bold relief, while the

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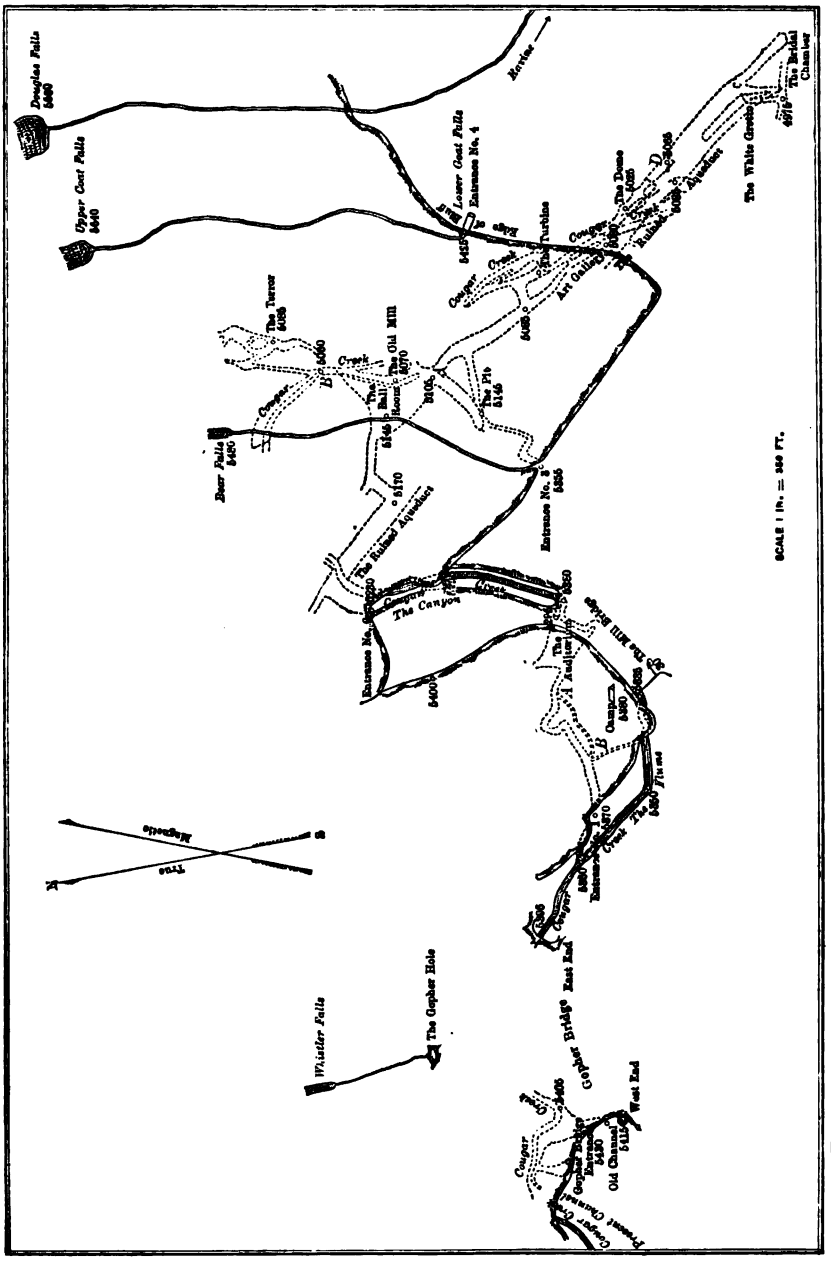

FIG. 1.—SKETCH-MAP OF DEUTSCHMAN'S CAVE, FROM A PRISMATIC COMPASS AND CLINOMETER SURVEY.

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marble between them has been eroded to a considerable depth. The geologist may here distinguish, much more clearly than in ordinary surface-exposures, the various stages of the metamorphosis of the original limestone into the present marble. There are evidences also of (comparatively small) cavities in the original limestone beds, formerly filled with white carbonate of lime, which was afterwards changed to marble during the metamorphosis of the limestone. Some chips and nodules of quartz, imbedded in these fillings, appear to have been carried into the cavities by water. In the gray and white marble, shrinkage-cracks are everywhere seen, which were formed during the early solidification of the limestone, and filled with carbonate of lime. They now appear as seams of white marble, usually at right-angles to the bedding of the rocks.

The creek, which now enters under Gopher bridge at the point marked "Present Channel," on Fig. 1, formerly had an entrance at "Old Channel" (now choked with drift), and also one at "Gopher Bridge Entrance," through which we descended into the cave. Westward from this point, "Grizzly" glacier, the source of Cougar creek, may be clearly seen. It derives its name from a grizzly bear which disputed Mr. Deutschman's right to invade his territory. The skin of this bear has been presented to me by Mr. Deutschman.

"Entrance No. 1" (Fig. 5) is 200 ft. down stream from Gopher bridge; and, 310 ft. below this opening, the creek plunges under a second natural bridge, 243 ft. long, called the "Mill" bridge, by reason of the roaring, as of many water-wheels, of the tumultuous stream beneath. The channel between Entrance No. 1 and the Mill bridge, which we named the "Flume," is cut in solid rock, and shows many pot-holes. For 160 ft. the descent is moderate; but for the next 150 ft. it follows the dip of the strata (30° E.), and the water plunges through a series of deep, connected pot-holes (Figs. 10 and 11).

At the east end of Mill bridge the creek emerges into the "Canyon," about 170 ft. deep and 234 ft. long, at the end of which it abruptly enters the cave by "Entrance No. 2."

Immediately east of the Canyon are three beautiful falls (Figs. 6, 7 and 8), named respectively "Douglas," "Upper Goat" and "Bear" falls—the first in honor of Mr. Howard Douglas, Superintendent of the Canadian National Parks.

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Fig. 9, taken from a point on Cougar creek about 1,000 ft. south of the cave, shows in the distance "Lower Goat" falls. At the foot of this cascade is "Entrance No. 4," through which all the water at once disappears into the cave. These falls can be seen from the railway, just west of the Loop.

Fig. 12 shows Cougar creek, in its passage beneath Gopher bridge.

Entrance No. 1 is the place where a part of the waters of Cougar creek sometimes enters the cave, particularly during very high water. Passing down this water-way, which follows the dip of the strata, the varied forms of the water-erosion are to be seen here to a better advantage than perhaps anywhere else in the cave. The walls have been fantastically carved by the torrents of snow-water that have rushed through it for centuries, and the channel is made up of a succession of rounded cistern-like cavities formed by the swirl and plunge of the water. The bottoms of these pot-holes still hold the boulders of quartzite that have ground them deeper and deeper; and the thin margins, where their walls have been worn through into adjacent holes, bear witness to the marvelous strength and hardness of the rock. The bands of white, mottled, and gray marble are shown in beautiful contrast, and, when wet, appear to be polished. This passage is from 4 to 10 ft. wide by from 10 to 30 ft. deep.

Fig. 13 shows the "Auditorium," containing some stalactites of ice and other ice accumulations, with Cougar creek in the center. It has an area of about 50 by 60 ft., and is about 150 ft. below the surface. From here the creek flows into the Canyon, 20 or 30 ft. away. The Auditorium will form the best pathway into the Canyon, when some débris has been cleared away. At present, to enter the Canyon from the surface would require a stairway more than 100 ft. long.

During our first exploration through Entrance No. 1, Cougar creek, suddenly rising and dividing its waters at the falls near the entrance, deluged this portion of the cave. We were drenched; our lights were put out, and we were obliged to abandon this passage-way for a time.

To get into the cave through Entrance No. 2, we descended almost vertically 85 ft. to the bottom of the Canyon by means of a rope. From the Canyon, the creek enters the largest of all

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the underground openings thus far discovered. It naturally should be the largest because of the accumulated waters traversing it. The average height, measured on the dip of the strata, is about 125 ft., and the width, measured perpendicular to the bedding-faces, ranges from 8 to 20 ft. The width varies because of the varying conditions of the flow of the water at the time of its formation. With all the water concentrated in one passage-way and flowing through it on a steep grade it would be narrow, and widest when on a moderate grade.

From the map it will be observed that those sections of the highest old water-way from Entrance No. 2 to the present southeasterly limit of the cave are all on a line, and that this line is coincident with the strike of the strata. The omitted sections of it have been explored sufficiently to determine that they are on the same line, but they are nearly filled with débris and are unattractive. The fact, however, that this old water-way, which we named the "Ruined Aqueduct," was originally continuous and straight along the strike of the strata, forms a base from which to study the subsequent changes. During its early history it undoubtedly appeared much like the passage-way in Entrance No. 1, but as the channel grew deeper and wider through erosion, many masses of rock from the hanging-wall were loosened and fell into the channel-way, thus causing an obstruction, around which the water cut its way, at the same time cutting away some or all of the obstruction itself. As a result, many enlarged places are to be seen here and there.

Still others are to be seen that have been formed as pot-holes, like round shafts, down which the water poured, keeping the boulders at their bottoms ceaselessly grinding them deeper and deeper. Under this process it was only a matter of time when, particularly at the confluence of streams, great masses of overhanging rock would be unfooted and dropped into the enlarged channel and pot-holes. This is shown to a marvelous degree where the waters of Bear falls formerly joined Cougar creek through "Entrance No. 3." Portions of the old channel-way and of the very large pot-holes, notably the "Pit," which is 120 ft. deep, are here visible, the other portions being covered with fallen rocks from the roof. One of these rocks, a very large one, rests in nearly a horizontal position, and its upper surface contains about 1,200 sq. ft. of floor-space. This we named

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FIG. 2.—COUGAR MOUNTAIN AND ILLECILLEWAET RIVER, VIEWED NORTHWESTWARD FROM THE "LOOP" ON THE CANADIAN PACIFIC RAILWAY.

FIG. 3.—VIEW SOUTHEASTWARD FROM THE CAVE. ROSS PEAK ON THE RIGHT; THE GREAT GLACIER ON THE LEFT; AND COUGAR CREEK IN THE CENTER.

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FIG. 4.—VIEW EASTWARD FROM THE "LOOP," SHOWING MOUNT SIR DONALD ON THE RIGHT.

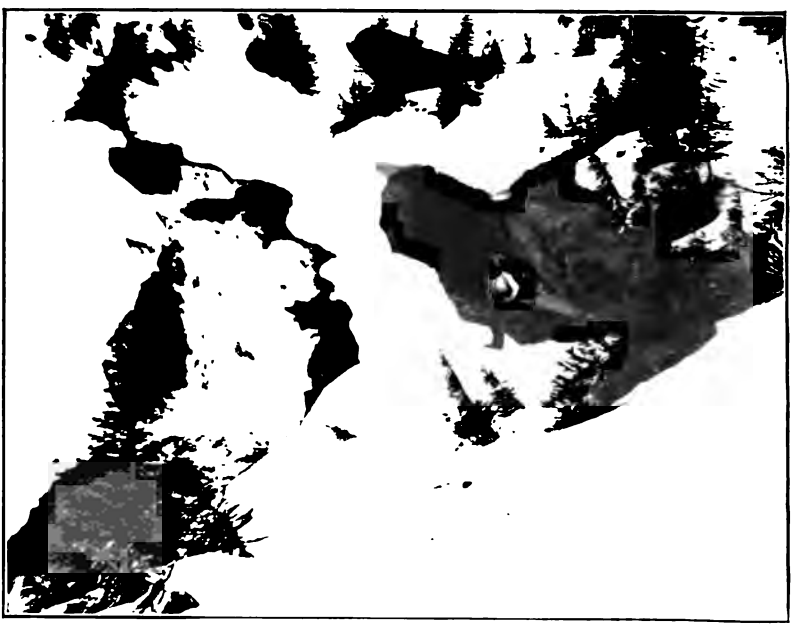

FIG. 5.—"ENTRANCE NO. 1" TO CAVE, SHOWING MR. DEUTSCHMAN, THE DISCOVERER.

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FIG. 6.—"DOUGLAS FALLS," EAST OF THE "CANYON."

FIG. 7.—"UPPER GOAT FALLS," EAST OF THE "CANYON."

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FIG. 8.—"BEAR FALLS," EAST OF THE "CANYON."

FIG. 9.—VIEW FROM COUGAR CREEK, 1,000 FT. SOUTH OF THE CAVE. "LOWER GOAT FALLS" IN THE DISTANCE, AT THE FOOT OF WHICH ALL THE WATER DISAPPEARS THROUGH "ENTRANCE NO. 4" INTO THE CAVE.

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FIG. 10.—POT-HOLES IN THE "FLUME," NEAR ENTRANCE UNDER THE "MILL BRIDGE."

FIG. 11.—POT-HOLES IN THE "FLUME" AT ENTRANCE UNDER THE "MILL BRIDGE."

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FIG. 12.—CHANNEL OF COUGAR CREEK UNDER "GOPHER BRIDGE."

FIG. 13.—THE "AUDITORIUM," WITH STALACTITES OF ICE, COUGAR CREEK IN THE CENTER.

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FIG. 16.—THE "BRIDAL CHAMBER." THE BED OF COUGAR CREEK IS BELOW, AND AT PRESENT INACCESSIBLE FROM THIS POINT.

FIG. 17.—VIEW IN THE SOUTHEAST END OF THE "RUINED AQUEDUCT," OVER THE "BRIDAL CHAMBER," SHOWING A BROWN ACCUMULATION OF LIME, SURROUNDED BY PURE WHITE.

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the "Ball-Room." The rocks in the Pit are of a very dark blue-gray color, and include bands of white marble, which have been crumpled by pressure to a zig-zag form.

The fallen masses of rock, wherever found throughout the cave, particularly those about the Ball-Room and the Pit, were carefully examined to determine their present stability. The roof was also examined carefully to the same end. The singular firmness of every fallen piece, even the very large ones, shows conclusively that the water had shifted all fallen pieces, great or small, into positions that are fixed and reliable. No evidence was discovered of any present movement in the roof, or of any places where the present water-erosion has made a fall of rock imminent.

At *A*, nearly opposite the Pit, we descended to the present bed of Cougar creek, at the bottom of the cave. This passage leads north directly under the Ball-Room, where the bottoms of several gigantic pot-holes, now in ruins, are to be seen. We naturally named this spot the "Old Mill." It certainly did grind for many centuries before it fell into ruin and disuse. Following up the creek to *B*, where it suddenly turns to the northwest, and continuing in a northerly direction, a series of chambers are to be found on the right of the passage-way. In the innermost of these are to be seen the most ragged walls that have been found anywhere in the cave. The jagged points and grotesque shapes at once inspire caution. This place we named the "Terror." The peculiar roughness is due to the partial metamorphosis of the rocks, and is similar to the condition found under Gopher bridge. In this case, however, the condition is accentuated by the existence of thin knife-like blades, instead of nodules, of unchanged limestone, all of which extend from 0.5 in. to 2 in. beyond the general surface of the marble holding them. The extreme south end of this inner chamber, which is 400 ft. below the surface, rises suddenly; and from its proximity to Bear falls, the inference is almost conclusive that it was at some time the inlet passage of the waters from those falls, yet it might as well have been the inlet from Upper Goat falls. Several other passages are also to be seen entering it from the north, near the roof; but they are all inaccessible, being nearly filled with gravel. They all enter it near the roof.

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Along this northerly passage, toward the chamber called the "Terror," the gravel in the bed of the channel is of a very different character from that in the bed of Cougar creek above the junction-point *B*. It is chiefly of a dark brown or reddish quartzite, while that found in the creek consists of marble and schist with occasional pieces of light or nearly white quartzite. This same dark brown quartzite had been observed at Bear falls, which corroborates the inference that the waters from these falls formerly entered the cave by this route.

This entire passage from the Terror to the Old Mill has been formed along a fault, which inclines upward at an angle of about 65° to the west. Along its line on the surface the ravine of Bear falls has been formed, and also the depression through which its waters now flow to Entrance No. 3. From this entrance these waters have cut their way down into the cave along the same fault, joining Cougar creek at the Old Mill, and in their passage have formed the Pit. This portion of the cave, just described as formed along the fault, is one of the most interesting and instructive sections thus far explored. It tells a long history from the first grinding at the Old Mill to the erosion of the present day.

Returning again to the point *A* and continuing along the passage-way, which from here runs in a southeasterly direction along the strike of the strata, many interesting features are met. The opening consists of a series of levels through which the water has successively carved its way, beginning with the Ruined Aqueduct, already described, and ending with the present bed of Cougar creek, 125 ft. below it. Among the most attractive spots are the "Turbine," the "Art Gallery," the "Bridal Chamber," the "Dome" and the "White Grotto."

At the Turbine, Cougar creek makes a reversed bend in the form of an overturned letter S. Immediately in the curve occurs a series of steep inclines and falls with a vertical drop of about 25 ft. Around this reversed bend and down these inclines and falls, the water, in the flush season, rushes and swirls with a deafening roar, which is greatly intensified by its reverberations in the cave.

Below the falls the course of Cougar creek is diagonally to the right across the strata, and its level is about 60 ft. below the passage-way. Just at the side of the present bed of Cougar

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creek, beginning just below the falls, is an old channel formed along the strike of the strata, from which are to be seen, looking up, two very large pot-holes, 18 ft. in diameter, and in a perfect state of preservation. One of them, with an arched roof about 40 ft. from the bottom, is decorated with carbonate of lime accumulations so delicate and dainty in effect that it might be the enchanted chamber of the fairies. This pot-hole we named the "Dome." All progress down along the course of Cougar creek from this point was barred by a very low roof at the side and by the steep descent and swift current of the creek itself.

Returning to the passage-way by which we were advancing, and proceeding southeasterly, we passed through an old water-course beautifully decorated with lime-accumulations, which we named the "Art Gallery." Fig. 14 shows the southeast end of it. Fig. 15 is a nearby view of a small section of the roof shown in Fig. 14. This lime-accumulation is white or creamy white, with an occasional tint of pink. It resembles heads of cauliflower set close together without any intervening spaces.

From the Art Gallery our course was down over large masses of fallen rock for a distance of several hundred feet to a narrow passage, continuing for another hundred feet, at the end of which we entered a beautiful room, which we called the "Bridal Chamber" (Fig. 16). The decorations of carbonate of lime are creamy white and are very dainty. This room is formed against a fault, and Cougar creek is here deflected by it to an easterly course. No way of getting down to the present bed of Cougar creek at this point, without ladders, was discovered. The roar of water plunging down a steep incline could be clearly heard, and it is assumed that the creek continues along this fault for some distance. We were greatly disappointed in not being able to descend to its bed, as this seems to be the only avenue of entrance to the openings that unquestionably exist between the Bridal Chamber and the place where Cougar creek emerges to the surface. Just where this last point is we have not been able to determine, as no outflow of water sufficient to locate it positively has thus far been found on the surface.

Immediately over, and about 125 ft. above the Bridal

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Chamber, is the extreme southeasterly end of the Ruined Aqueduct. Fig. 17, from a photograph taken in that section of it which lies above the Bridal Chamber, shows a beautiful brown accumulation of lime surrounded by a drapery of pure white.

A small cave exists directly over Entrance No. 2, at the north end of the Canyon, which has been explored and named the "Ice Cave," because a part of the winter's accumulation of ice remains in it during the entire summer.

The Canyon and the Ice Cave have been formed along a fault, nearly or quite parallel to the fault previously described as extending from the Terror to the Old Mill. The parallelism of these two faults and of the beds of the streams leading from Upper Goat and Douglas falls, gives a hint at the possibilities of finding very extensive openings along the faults that probably formed the beds of these streams, similar to the open-ings found along the fault that formed the ravine of Bear [illegible].

As already remarked, the waters from Lower Goat falls immediately disappear into the cave at the foot of the falls. They then flow underground northerly for a few hundred feet, and then descend nearly vertically into unknown cavities below. Just where the stream from these falls joins Cougar creek has not been determined, but it is quite probable that the junction is beyond the Bridal Chamber in a southeasterly direction, since no branch passage has been discovered, between the Old Mill and the Bridal Chamber, corresponding in size with the entrance at the foot of the falls.

1. *The Extent of the Cave.*

The total length of the passage-ways I have surveyed and measured thus far amounts to about 4,000 ft. What lies beyond the Bridal Chamber, between it and the place where Cougar creek comes to the surface, and also what exists along the faults which have formed the beds of the streams from Douglas and Upper Goat falls, is entirely unknown; but great possibilities are suggested as to the existence of caverns even larger and more beautiful than any thus far explored. One suggestion seems almost certain—namely, that beyond the Bridal Chamber the openings must be at least as great as those between this point and Entrance No. 2. In fact, this should be

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the largest part of the cave, by reason of its being a continuation of the portion already explored, with added waters to aid Cougar creek in forming it.

2. *The Probable Age of the Cave.*

Notwithstanding the fact that the rocks belong to a comparatively old series, the beginning of the cave undoubtedly dates from recent geological time. Any attempt to estimate its age by assuming a rate of erosion would be mere guesswork. Actual rates of erosion for one locality or one kind of rock would not apply to this particular case, even though such data were at hand. Therefore, no intelligent estimate can be made until the present rate of erosion has been determined. In my report to Mr. Douglas for the Canadian government, I have suggested the value of such a determination, and have pointed out that "in several places along Cougar creek, in the bottom of the cave, an excellent opportunity is afforded to determine accurately the present annual rate of erosion. A micrometer measuring-apparatus should be used, and accurate computations made of the area of the cross-section of the rock eroded per year; also of the quantity of water passing a given section in one year, and its velocity. From these data the ratio between the area of the cross-section of the average stream and the area of the cross-section of the rock eroded could be determined. In other words, the volume and velocity of water required to remove the rock eroded in a year could be found." The nature and quantity of the materials carried by the water should be determined and measured at the same time and with the same degree of accuracy, for the reason that the sand, gravel and other materials thus carried form the eroding instruments of the water, and are made effective by its velocity and momentum. These cross-section measurements should be made at a sufficient number of places to determine the rate of erosion under different velocities of the current, for it must be borne in mind that where the sand and gravel lodge in the bed of the stream, the rate of erosion is many times smaller than where the bed is continually swept clean by a more rapid current.

On the assumption of $\frac{1}{32}$ in. a year, it would have required 48,000 years to erode the 125 ft. of depth of opening found in the cave.

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3. *Has the Cave Ever Been Inhabited?*

No evidence whatever was discovered that any portion of the cave has ever been inhabited by human beings or wild animals. The continued rapid circulation of air through the cave causes ice to form in great quantities in some parts during the winter, and lowers the temperature throughout the cave to such a degree that it would not be an attractive home for man or beast. In addition to the low temperature in winter, the thunder and roar of the cataracts in the cave during the spring and summer, particularly while the high water exists, would make it a highly disagreeable habitation.

POSTSCRIPT.—Since this paper was presented, information has been received that the Canadian government, during the past summer, has not only constructed a good trail to the Cave, but also has built three cabins, one as a residence for Mr. Deutschman, who is to be the care-taker, and the other two as shelters for visitors, both men and women. The excursion to the Cave can therefore now be made without danger or hardship by tourists of all classes; and, since the Canadian Pacific Railway Co. intends to advertise it widely as one of the attractions of the Selkirks, it will soon become known to thousands who now hear of it for the first time.

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The Corrosion of Water-Jackets of Copper Blast-Furnaces.

BY GEORGE B. LEE, DOUGLAS, ARIZ.

(Toronto Meeting, July, 1907.)

DURING the two years in which the new reduction-works of the Copper Queen Consolidated Mining Co. have been in operation at Douglas, Ariz., there has developed a remarkable condition in regard to the corrosive action of the water used to cool the jackets of the blast-furnaces.

Were it not for the many contradictory features, it might pass as one of the unavoidable troubles due to the composition of the water. This water, obtained from wells 600 ft. deep, is also used in the steam-boilers, and its composition, as shown by the following analysis, does not indicate the presence of any ingredient which would explain the corrosion:

	Grains Per U. S. Gal.
Silica,	0.861
Iron oxide and alumina,	0.223
Calcium carbonate,	0.261
Calcium sulphate,	.....
Magnesium carbonate,	.....
Sodium and potassium sulphates,	14.850
Sodium and potassium chlorides,	9.732
Sodium and potassium carbonate,	6.482
	32.409

The jackets are made of inner plates 0.5 in. thick and an outer plate $\frac{3}{8}$ in. thick, with $\frac{3}{8}$-in. stiffeners between the inner and outer plates. In from 10 to 12 months the inner plates have been reduced by corrosion to a thickness of from $\frac{1}{8}$ to $\frac{1}{16}$ in., while the outer plate in the same time is reduced by less than $\frac{1}{16}$ in., and the stiffeners show very little corrosion. The plates are pitted and eaten away in some places more than in others. There is practically no scale found on the jackets, but when cleaned considerable iron oxide is found in the bottoms.

With an action so marked, serious trouble would be expected in the boilers, but, on the contrary, a recent inspection by a

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boiler insurance company gave an almost perfect report on the large boiler-plant, which consists of eight 500-h.p. Sterling boilers. There was no pitting in the tubes. The inspector's attention was particularly called to the pitting of the jackets. The cast-iron impellers of rotary pumps that pump to the cooling-tower from the hot-well are pitted in spots quite as deeply as the jackets. A sheet-iron pipe 0.25 in. thick that has carried all the hot water for the jackets of eight furnaces has never leaked. Some wrought-iron pipes, handling water at a temperature of from 65° to 80° F., have been almost destroyed by pitting, while others in the same line have not shown a leak.

These notes are offered in the hope that some member of the Institute may have met this problem before, and can throw light on this interesting subject.

Discussion.

William Kent, Syracuse, N. Y. (communication to the Secretary*):—The analysis of the water shows it to be somewhat unusual; it is rather high in sodium and potassium sulphates and chlorides, 24.58 grains per gal., and very low in calcium carbonate, 0.261 grain per gal. There are three theories which may account for the corrosion:

1. *Air-Bubbles Lodging on the Iron.*—It is well known that even pure water, such as the water of condensation from steam heating-systems, is an active agent in causing the pitting of the nipples used for connecting cast-iron radiators and the iron or steel return water-pipes, and the presence of air in the water is supposed to be the real cause of the corrosion.

2. *Electrolytic Action.*—The water containing sulphates and chlorides may act as an electrolyte, and different portions of the steel plate, having slight variations in chemical composition, may act as two different metals or electrodes.[1]

3. *Chemical Action.*—At certain temperatures potassium sulphate may attack iron, forming iron and potassium sulphate.

Possibly all three of these actions may take place at the same time.

The remedy indicated is the addition of a little milk of lime

* Received July 17, 1907.

[1] See The Corrosion of Iron, Dr. Allerton S. Cushman, *Proceedings of the American Society for Testing Materials*, vol. vii., pp. 211 to 228 (1907).

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to the water. This will neutralize any acid reaction of the potassium sulphate, and form a precipitate of calcium sulphate, which will make a protective coating on the iron and prevent all three of the actions above described.

JAMES DOUGLAS, New York, N. Y. (communication to the Secretary*):—The following extracts from correspondence with Mr. Lee give some additional particulars concerning the corrosion of the jackets:

DOUGLAS, ARIZ., June 12, 1906.

We have just taken out from one of our new furnaces a jacket, which has been in use five months. I find the same trouble as before: the inner sheet is very badly pitted. The outer sheet and the angles that space the inner and outer sheets do not appear to be attacked.

In connection with this, I wish to call attention to the fact that our boilers, which have been in use now for two years, have just been inspected by the insurance company and have received an almost perfect clearance. Apparently the corrosion is due to some peculiar condition that exists with the fire on one side and water on the other; and also that there is a difference between this condition and that obtaining in the boilers. The steel plate on the outside, which is much thinner to start with, and which is air-cooled on one side, with hot water on the other, far outlasts the thicker inner sheet, and the T-irons, or angles, which are immersed in the water between the two sheets, seem to be very little attacked.

In connection with this peculiar action, I would call attention to the impellers in the rotary pumps which circulate the water for the condensers in the power-plant. These are made of cast-iron; and we find that at certain points they are very badly pitted, being eaten away to a depth of a quarter of an inch for a space of one to two inches in area; and right next to this there will be spaces that are apparently not attacked at all. We have had whole lengths of pipe, leading from the supply-tanks to the power-house, which were perforated, and lengths next to them apparently very little attacked. The surface of the jacket seems to be more uniformly attacked, but even on this there are smooth spots that have apparently resisted this action.

You will see from the above that materials as different as flange-steel and cast-iron are both attacked by the water; that steel exposed to very hot water, such as exists in the boilers, is apparently not attacked; that pipes handling water not over 75° F. are attacked; that pumps handling water both cold and moderately heated are attacked; and that steel surfaces heated on one side and with water on the opposite side of, say, 140 to 150° F., are badly attacked, while plates cooled on one side by the air and exposed to the same water, are very little attacked.

I suggested that samples of the inner shell and a stay-bolt be sent for inspection by the members of the Institute, and that the temperature of the water in contact with the inner and the

* Received Sept. 28, 1907.

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outer shell be taken. To this request the following reply, dated Dec. 11, 1906, was made:

Complying with your request, I am sending to-day, by express, a stay-bolt and piece of metal from a water-jacket. The temperatures which you suggested taking of the jacket-water near the inner sheet and near the outer sheet have been taken repeatedly, and I inclose a statement showing the range of temperature. You will note that, as the temperature increases, the variation also increases. We have just now a report from the inspector of the Hartford Boiler Insurance Co., in which we are given an absolutely clear bill on the entire battery of boilers, six of which have been in use 3½ years.

I believe I told you on your last trip here that we are now making experiments by feeding oil into the water as it goes to the jackets, with the hope of coating the inside of the jacket with a film of oil, and possibly preventing the corrosive action of the water.

Temperature of Water in Jackets Near the Inner or Fire Side and Outer or Air Side.

Outside. Degrees F.	Inside. Degrees F.	Outside. Degrees F.	Inside. Degrees F.
97	106	104	115
98	107	106	115
98	108	108	118
102	112	116	130

In answer to a request as to the effect of this oil, Mr. Lee wrote Aug. 28, 1907, as follows:

I am in receipt of your wire of the 22d in regard to corrosion of water-jackets. Apparently the use of oil has been of some benefit in reducing this corrosion, as the amount of jackets renewed seems to be little more than it was, though the number of jackets in use is considerably larger. I recently had occasion to examine a jacket which had been taken out, and found a very peculiar condition: namely, there was a place about 6 by 8 in. right in the middle of the plate which seemed to be quite smooth and not pitted at all, while all around it was very badly corroded. This jacket had been in use about a year and a half.

I have received a very interesting letter from Mr. Beardsley, who was formerly with the Mount Lyell Co. in Tasmania, giving me a number of experiences that he had had of corrosion of jackets, and, as far as he was able, the causes. In one instance well-water was substituted for the former supply. The well-water seemed to be highly charged with gas, and they experienced great difficulty from the jackets burning. This he attributed to the formation of gas-bubbles on the fire-sheet. By mixing the city water and this water the trouble was very much reduced, and a return to the city water stopped it entirely. This, of course, is quite a different experience from ours, which is not one of burning, but one of interior corrosion.

The following letter, dated London, Sept. 17, 1907, is from George M. Douglas, a member of our staff, who was an engi-

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neer for some time on the White Star and other steamship lines:

On reading the correspondence you have received from Mr. Lee at Douglas regarding the corrosion of the inner plates of the furnace-jackets, I suggest that this might perhaps be caused by some electrolytic action. This same corrosion takes place in the Scotch type of marine boiler, particularly when the water contains a little salt. This boiler is somewhat analogous in form with the jacket, having a hot inner plate, a water space, and relatively cool external shell.

This corrosion is prevented by hanging zinc plates on the stay-rods between the spaces affected. It is also a practice to put zinc plates near the water-inlet, so that any free acid in the entering water may combine with the zinc and be neutralized.

Perhaps a similar application of zinc to the jacket-shells at Douglas might prove beneficial. I suggest applying it in the following manner: The zinc plates should be about $\frac{3}{4}$ in. thick, about 8 in. wide, and 16 in. long. Some authorities object to the application of zinc direct to the iron, though it is customary to do so in British engineering practice.

A suitable means would be to have brackets made of copper strips, $\frac{3}{8}$ in. by 1.5 in., placed about a foot above the bottom of the jacket on the inner shell, into which brackets the plate could be inserted from the top. I also suggest that a plate be put where the jacket-water enters. Two plates of the size mentioned on each side and one on each end of jacket (should there be any corrosion there) is enough. I do not know what the size of the furnaces is. But the relative proportion of zinc surface to iron surface should be about one to ten. A good contact should be insured between the zinc and the copper and iron, or, if copper is not used, between the zinc and iron.

It is not enough merely to place the plates in the water space; they should be well fastened to the jacket.

Under the same date, Mr. Douglas, in response to an inquiry from me, made, in substance, the following statement, which, although not directly pertinent to the present discussion, may be valuable as a contribution to the general question of the corrosion of steel and iron:

With regard to the corrosion of stern-posts and plates in the vicinity of propellers on ships, I would say that this action seems to be well understood as due to the fact that the propeller-blades, etc., on one hand, and the stern-posts and plates, on the other hand, are of different material, so that a galvanic action is set up, the salt water acting as an electrolyte. If all these parts were of exactly the same material, no corrosion would take place; but this is not the case in practice. The stern-tube is usually of bronze, and the propeller of bronze or steel, with blades of manganese bronze. An interesting case is on record, of a vessel on which iron propeller-blades were replaced by blades of bronze. Immediately upon this change, the corrosion of the stern-post and surrounding plates became so great they had to be renewed after only one voyage to the Cape. They were afterwards protected by zinc sheathing; and it is now the custom to protect such parts by sheathing of zinc or some metal of similar electro-chemical character.

Though these facts are interesting, I fear they will not help you much in deal-

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ing with your jackets, since the conditions of your problem are by no means the same as those of the marine practice above stated, in which both the origin and the remedy of the trouble seem to be clearly established.

With regard to the general question of the corrosion of steel or iron plates, however, I may call your attention to one point which may be worthy of consideration—namely, the electro-chemical relations between metals and their oxides. According to a leading author,[2] "every metal is electro-positive to its own oxide." When steel or wrought-iron, with oxide scale upon it, is placed in an oxidizing liquid, the conditions of active corrosion are complete; and even without a specially oxidizing medium, it is asserted that a galvanic action may be set up in the presence of air and moisture between the metal and its scale. It is therefore regarded as very important that no "black oxide" should be left on the plate; for, though in itself it tends to protect the surface (the black or magnetic oxide of iron resisting ordinary oxidizing agents), yet if, in finishing, handling, or subsequently using the plates, portions of it should be knocked off, the remaining portions contribute to the corrosion of the exposed metal.

In 1879, Sir Nathan Burnaby declared, as the result of his observation, that when mill-scale was left upon plates and angles used in ships, its effect upon neighboring surfaces of bare metal was as strong and continuous as that of copper.

In 1882, Mr. Farquharson conducted for the British Admiralty, at different naval stations, exhaustive experiments as to the action of mill-scale on ships' metal exposed to the conditions of marine use, and found: (1) that no "pitting" occurred in mild steel *freed from all scale;* (2) that the loss of weight by corrosion was practically the same for clean mild steel and clean iron; and (3) that the action of mill-scale in inducing corrosion is considerable and continuous—equal in these respects to that of an equal amount of copper.

The Admiralty practice is to pickle all ships' metal, for the removal of mill-scale. The scale may also be removed by the sand-blast, or by means of a gasoline-torch, followed with a scraper and a wire brush. Pickling, however (in dilute sulphuric or hydrochloric acid), is probably more thoroughly effective.

The rivets should be of the same material as the plates. Iron rivets in steel plates might cause trouble.

Hiram W. Hixon, Victoria Mines, Ontario, Can. (communication to the Secretary *):—I have had difficulties here similar to those encountered at Douglas, and I found the cause to be the carbonic acid given off when the water was warmed. All the water in the streams in this country contains organic matter coming from peat-bogs and muskegs. It is brown in color, and when it strikes the fire-sheets of the jackets the carbonic acid is given off and travels up along these fire-sheets because of the bosh in the furnace. The lower side of the tuyeres was

[2] *Metallic Structures: Corrosion and Fouling, and Their Prevention*, by John Newman, p. 36. London: Spon & Chamberlain (1896). See also, *Rustless Coatings, Corrosion and Electrolysis of Iron and Steel*, by M. P. Wood. London: Chapman & Hall (1904).

* Received through Dr. Douglas, Sept. 28, 1907.

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much pitted, and they leaked badly until I had copper tubes put in in place of iron ones.

The inner or fire-sheets were destroyed most rapidly opposite the cold-water inlet, where the greatest amount of carbonic acid was given off. Our boilers are not affected and are perfectly clear of scale. I think the acid is liberated in the feed-water heater, in which there are copper tubes, and after it is in a gaseous condition it does not attack iron, or at least the water is necessary to make it destructive. The pipes leading from the feed-water heater to the boilers are destroyed, but the boilers are not.

The Canadian Copper Co. had a purifying plant for the feed-water, and the pipe leading from the purifier to the different boilers went over the boilers, and each lead to the boiler came out of the bottom of the main pipe. Tests made of the water to the different boilers showed that the water to the boilers nearest the purifier was much less acid than the water to the boiler at the end of the feed-pipe. The superintendent spoke to me about it, and I suggested that the acidity of the water was due to carbonic acid dissolved in the water, and that being a gas it had a tendency to enrich the water in the top of the feed-pipe, and, consequently, the water drawn off for the first boiler from the bottom of the main contained less acid than the water which went to the last boiler.

I think the trouble at Douglas is due to the water-supply coming from the deep wells containing carbonic acid, and this acid is probably due to the source of the water being something in the nature of a mineral spring, such as Saratoga, Manatau or Apollinaris. Ordinary chemical tests would fail to detect any mineral acid, and the gas being small in quantity would escape detection.

The remedy for the trouble is to use copper fire-sheets, or to run the water through cooling-towers and use it after the carbonic acid has escaped.

C. D. Van Arsdale, New York, N. Y. (communication to the Secretary*):—There are several explanations which present themselves regarding the corrosion of the water-jackets of the

* Received Oct. 31, 1907.

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Douglas furnaces. The most obvious of these—namely, that the composition of the water is itself directly responsible—may be dismissed as improbable. Analysis of the water shows that it may be called a good boiler-water for this region, since it contains very small amounts of incrusting solids and the non-incrusting solids are not excessive; and this opinion is verified by its causing practically no boiler-troubles. Since no corrosion takes place in the boilers, it is evident that the dissolved constituents of the water do not alone afford sufficient explanation of the trouble.

Granting that the water is itself non-corrosive, there is nothing in the working of the ordinary water-jacket to account for the difficulty, otherwise such corrosion would be more or less generally observed in other plants. It would, therefore, seem that the only explanation left is electrolytic action; but it is not evident what is the cause for the electrolysis.

It is well known that lack of uniformity in the composition of iron will cause corrosion on account of action due to minute local galvanic couples. If this is the cause, then a suitable remedy would be to hang zinc sheets inside the jackets, as has already been suggested. Another cause of electrolytic corrosion may be stray currents from some source. In the same way much trouble has been experienced from corrosion of gas- and water-mains in cities, due to stray electric currents passing along them. A very small current has been found sufficient to cause a great amount of trouble, but if this should be found to be what is causing the electrolytic action in the jackets, it should be quite simple to put a stop to it.

The different temperatures observed in different parts of the jackets might also be sufficient to cause some corrosion, since electrical currents can be produced in an electrolyte by electrodes of the same metal, portions of which are at different temperatures. This could be obviated by a circulation of water in the jackets sufficiently rapid to do away with any differences of temperature.

The fact that the jackets are much more corroded on the fire side seems to indicate that the electrolytic action is due not to lack of uniformity of the iron, but to one or both of the other causes mentioned.

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DISCUSSIONS.

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Blast-Furnace Practice.*

BY T. F. WITHERBEE, DURANGO, MEXICO.

Discussion of the paper of F. L. Grammer, Flue-Dirt and Top-Pressure in Iron Blast-Furnaces: A Study of the Influences Controlling Them, *Trans.*, xxxiv., 92 to 105; and the paper of J. E. Johnson, Jr., Notes on the Physical Action of the Blast-Furnace, *Trans.*, xxxvi., 454 to 488.

THE matter of blast-penetration is merely a question of blast-velocity through the tuyeres, and when the distribution of the charge admits of no variation or control, as is the case with ordinary double and single bells, it involves merely that the adjustment of total tuyere-area to blast-volume shall be determined for average conditions by direct experiment and maintained by slight changes of blast-volume. Penetration is perhaps the most important factor connected with the practical blowing of a furnace.

When proper penetration is once attained the relation of tuyere-area to blast-volume admits of change only to a very limited extent. I can testify to the value of probing with an iron (not a steel) rod, as described by Mr. Johnson, to ascertain the conditions at the tuyeres, and I would add that it is about as useful considerably higher up. However, its reading is not quite so simple as might be supposed, as it may deceive if judged by its temperature (color) alone, and there should also be taken into account the location of the hard and soft zones, if any, when passing the rod across the crucible, for which reason the furnace-manager or the one who "wants to know" should assist in the probing.

If the rod should show the crucible to be hot at the walls

* In view of the fact that the authors of contributions to discussions in this department habitually comment upon more than one preceding paper, so that it is impracticable to class such a contribution as a discussion of any particular paper alone, it has been deemed best to use the general heading "Blast-Furnace Practice," as was done for the same reason in former volumes of the *Transactions*.

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and cold in the center, and also develop a soft center, or even a cavity, it is *prima facie* evidence that the furnace is already overblown.

In such a case of doubt it is best to "blow up" for half an hour or so, and then test again, when if found abnormally hot at the center, or hotter than at the wall, overblowing has occurred. Quite often, when every other symptom has indicated the advisability of a temporary slackening of the blast, the test-rod has unmistakably demanded more blast. While in the practice to blow a constant volume of blast without watching and controlling penetration, it may well be that some of the derangements are due to such neglect. Some may object to any change of blast-volume, and it may not be so much a necessity with Bessemer and basic furnaces, especially in the case of a group of furnaces having access to a "mixer" to average up the quality of the iron; but at isolated furnaces, especially foundry-iron furnaces, restricted to only three grades, with no outlet for "off-iron" except the bargain-counter, slight changes in blast-volume offer certainly the promptest and probably the best remedy available at present: while the use of charging-apparatus having controllable distribution and mixers may be the better remedy of the future. Had I known the value of the test-rod way back in the 70's and 80's I am sure I should have escaped 90 per cent. of all the troubles that occurred, and incidentally should have missed some "experience" and "practice," which, while not entirely devoid of value, cost somebody more than it was worth. It is evident that testing with a rod has been independently taken up by many furnace-men. Mr. Edgar S. Cook told me at the Monterey (Mexico) meeting of the Institute, that he had used it, presumably during his active furnace-practice, which would date back some considerable time. My experience with it began in December, 1895, after having discovered a "core" in the South Chicago Calumet furnace by other means.

In watching inside-conditions with an iron rod, while removing the core, the value of the test-rod at once became evident. I consider it the most valuable piece of "apparatus" available to the furnace-manager known to-day, and Mr. Johnson will doubtless receive the thanks of all who were not familiar with it. It would seem to be entirely a matter of choice whether

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"top-pressure" exists at all or not, as the same gases that may have been under pressure at the furnace-top are afterwards burned at stoves and boilers under the conditions of a partial vacuum, notwithstanding their greatly increased volume due to the addition of the air of combustion and their expansion by combustion, finally passing into the chimney at about furnace-top temperature; and it is only necessary to provide adequate down-comer flue- and valve-area and sufficient chimney-draft to eliminate it entirely. However, for reasons given later by the late Mr. Thomas Whitwell, some top- or flue-pressure should be maintained, but such pressure should be had by gas-valve adjustment at stoves and boilers.

Any top-pressure (9 in. or thereabout of water), as noted by Mr. Grammer and Mr. Firmstone, was probably due to inadequate down-comer flues and valves, or to the same obstructed by flue-dust. But, whatever the cause, such pressure is of no use, may be positively harmful, and should be avoided.

An instance of the destructive effect of gas-pressure occurred at a coke-furnace using Lake ores. The gas in the Spearman burners burned with a series of puffs and explosions, with a loud drumming noise, and had set up a shaking, swaying movement of the stoves, amounting to more than 0.5 in. at the top, with the result that at about three-quarters down the checker-work the bricks were ground to powder, the checker-work settling down from the top into a heap of loose bricks. The shaking was at once recognized as due to the puffy combustion, from the same thing once having occurred with an iron-pipe stove, though from a different cause. As the pressure was the only abnormal condition known, the cause of the explosive combustion was sought in that direction. The pressure was found to be as follows: At top, 9 in.; at outlet of first dust-catcher, 8 in.; at second dust-catcher, 8 in.; at outlet of second dust-catcher, 2.5 in.; and at all burners, 0.5 in. of water.

By removing the diaphragm of the second dust-catcher, the pressure fell to 2.5 in. all around, except at the top, where it was 3.5. in., the gas burned quietly, and all shaking at stoves ceased.

An instance of high gas-pressure due to obstructed flues also happened at a coke-furnace using very poorly-made coke. The tar condensed in the dust-catcher goose-neck, and cemented

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the coke-dust as it came along, until it had closed the flue to about 12 in. Then came some large coke and choked it so much that the loaded bell would not open. The holding-up of the bell was thought to be due to the tar actually cementing it to the hopper-ring, but finally it was noticed that it would dump when the blast was off, which located the difficulty. That has occurred three times, giving no sign until it became an accomplished fact.

Abnormal pressure is probably due, in some cases, to too-small down-comers which have not been based on any consideration of the volume of gas to be carried. This I infer from a comparison of a large number of furnaces which show no harmony as to flue and down-comer areas. Many years ago, Truran laid down the rule of one-sixth of the stock-line area, and such rule applied now would call for an increased size at many furnaces, but our blast-furnace designers doubtless now have something more modern and exact.

As to what flue-pressure should be, the late Mr. Thomas Whitwell's advice as to handling gas in connection with his stoves may be as good now as when given 30 years ago. He said: "Always keep a little pressure in the gas-flue, just enough so that there is a faint smell of unburned gas, and then you will know that it is not burning in the flues." Probably about 3 in. of water-pressure would be ample.

I do not know who Mr. Johnson aimed at in his discussion of "blast-wandering." I used the phrase in my paper on special forms of charging-apparatus,[1] though the expression was not original with me, having been given me by a furnace-man of 35 years' experience, beginning with anthracite coal and magnetic ore and ending with a group or plant of 18 or 19 blast-furnaces using coke and Lake ores, whose fuel economy and regularity of working have never been equaled, so far as I know—a man naturally conservative, who never announces anything on furnace-practice that is not based on repeatedly observed facts, and as I most fully agree with him, also on account of observed facts, it seems proper to defend the claim. But, first of all, it will be necessary to state what was meant by "blast-wandering," and on that point I can speak for only my-

[1] *Trans.*, xxxv., 575 to 586 (1905).

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self. As used in the paper referred to, a careful reading of it will show that only wandering through the tuyeres was referred to, and of which I believe indisputable evidence was cited in the fact that some tuyere-pipes were 400° or 500° F. hotter than others, and also the device of Mr. Hartman, which showed a difference of pressure in different tuyere-pipes; and I also might have mentioned the experiment of another furnace-man who introduced a small turbine-wheel into the belly-pipes, the shaft of which projected out through a bushing in the peep-hole. The different numbers of revolutions shown at different tuyeres were assumed to be due to differences in blast-volume passing into the furnace, and therefore to demonstrate blast-wandering. Now, if there is any other explanation for the above phenomena than blast-wandering, it would be interesting to hear it. Blast-wandering, as understood by myself, means nothing more or less than following lines of least resistance, and that does not necessarily imply that such lines are produced by the blast, but that the blast takes advantage of them and so at times does great damage, and in that view the burning through of the linings at Sydney, as related by Mr. Baker,[2] the case cited by Mr. Porter,[3] and my own experience at Mayville, are all cases in point. In old anthracite practice it was a common saying: "We are working a lot of little furnaces inside of a big one," meaning that the tuyeres were working somewhat independently of each other. Of blast-wandering in the shaft of a blast-furnace I can cite but a single example—that of a furnace that had been scaffolded by alternate under- and over-blowing, while using all coke. To remove the scaffold, six holes had been cut through the shell and lining 35 ft. up, or just half the height of the stack. After normal conditions were restored, five of the holes were solidly bricked up and one was only loosely stopped by bricks, backed up by dry sand, so that it could be easily opened to see what was going on inside. Sometimes the stock at the walls was at a bright red, and was moving freely downwards, and there was a free escape of gas, but never enough to indicate much pressure. At other times all would be black, with only sluggish movement of materials, and no gas escape, a rod showing a hot center, and a center de-

[2] *Trans.*, xxxv., 244 to 255 (1905).

[3] *Trans.*, xxxv., 1017 (1905).

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scent of the charges. All this taking place without any change in the charging or blowing. Red-hot tuyere-pipes and a few holes cut through the shell and the lining between the mantel and the stock-line might furnish some valuable information, even at the present time. I will add that channels of least resistance are formed by any kind of bell-charging—at the inner and outer edges of the inverted V ring of materials in case of a single bell, and, in addition, in case of a double bell, a channel between an outer ring of material and a center heap, if the double bell has that kind of distribution—by reason of the coarser part of the charge rolling from the apexes and so separating from the finer part, and I have had abundant evidence that such rings and heaps descend very low in the furnace practically intact.

During the last two years I have been experimenting with a new form of bell, which gives controllable distribution, consisting of one bell on top of another, with ports or notches below the lip-ring seat. When the ports are closed the distribution is that of a single bell, but when the ports register with each other, or are open, the distribution is three-fold—viz., a segment of a large ring, a segment of a smaller ring, and a center heap, and as the bells make $\frac{3}{25}$ of a revolution on a vertical axis each time they close, a spiral, stratified distribution is produced. By proper manipulation a hard or a soft center can be produced at the tuyeres. Each port has a deflecting wing, and by varying the inclination, shape and angle of the same, the charge can be given any distribution required. This bell has led to 20 per cent. increase of blast-volume and 10 per cent. increase of ore-charge, giving 32 per cent. more product.

The general effect with this bell is to break up the channels, just as it is necessary to do when washing gelatinous precipitates in chemical analysis.

How gas-pressure decreases from the tuyeres upwards can be easily determined by experiment, but according to Mr. Johnson's theory, that the pressure and ascent are equally distributed over any given horizontal section, it should fall over 60 per cent. at the bosh-top of an 18-ft. furnace with an 11-ft. crucible, or to less than 6 lb. in the case he cited.

Mr. Fred H. Foote gave me a couple of pointers on "explosions" which I have never seen brought out in papers, or elsewhere, that "explosions seemed to occur with most frequency

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in furnaces that were running very close on fuel, and that there appeared to be something of a warning by the appearance of that very low silicon iron, which is covered by that well-known brownish powder upon cooling, consisting mostly of carbon and silica, which can literally be seen to exude or boil out of the pigs." Explosions which began at a coke-furnace in the spring of the year continued with increasing frequency and violence until common sense suggested lightening the burden, when the trouble ceased.

Afterwards, Mr. Foote told me it was necessary to take off from 200 to 2,000 lb. from about a 16,000-lb. ore-charge in order to keep furnaces in that locality on an even keel, when the moist season came. As it sometimes occurs that furnaces burn out the center very high up, even clear to the bell, it would seem that ignition of carbon-dust might be responsible for some of the "blows" that Mr. Johnson attributes simply to pockets of gas under pressure. Hanging and slips, at one time, were attributed in some cases to the furnace having worked up "too hot." I would call attention to some experience in changing from foundry to high-silicon iron, which required taking from 750 to 800 lb. of ore from an 8,750-lb. burden, leaving the charge for high-silicon iron 8,000 lb., or just a 2 to 1 burden, the coke-charge being 4,000 lb., and raising the blast-temperature from 1,000° to 1,300° F.

Now that was quite a reduction in burden, especially as the furnace was running very hot on foundry-iron, and it seems that all the conditions were provided for hanging and slipping, as the furnace appeared to be about as hot as fuel and hot blast could make it, yet slips and hanging were very rare, and, in fact, on that grade of iron were practically unknown, for which reason it has occurred to me that the theory would fit better if turned end for end—*i.e.*, that a furnace "works hot (sometimes) because it hangs" instead of "hangs because it is too hot."

The proposal to give the shaft of a furnace a more rapid taper from the stock-line down is, like the now pretty general lower and flatter bosh, a return to old charcoal-furnace lines of from 50 to 75 years ago. As the more rapid taper would naturally accompany a smaller diameter of throat, it should be an improvement, since it would force a better stock-distribution and locate the charge nearer the center, in a degree plugging

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up that easy route and compelling a breaking-up of a ring of impervious fine materials, due to that most efficient device for its formation—the single charging-bell.

Mr. Uehling once stated, in a discussion, that in his opinion 90 per cent. (I believe) of all blast-furnace troubles were due to faulty charging-apparatus, and I firmly believe he did not overstate the case.

Before a slip or "blow-up" can possibly occur a cavity must be formed or burned out for the materials to slip into. The single bell always distributes the charge in the same way, usually leaving a loose center, and it burns out, if not prevented by probing and changes of blast-volume, and so the stock adheres to the walls, or it arches over, using the walls as skew-backs, and the result may finally be a slip of more or less violence. On the contrary, a furnace-charge placed well in the center is naturally more self-sustaining, and under such conditions slips less frequently.

I never saw any call for the complaints against Mesabi or other very fine ores, due to fineness, though it is probable that the extreme fineness made that impervious ring more dense, and so accentuated an already existing trouble.

For it must not be overlooked that furnaces have always slipped, and long before the Mesabi ores were discovered. Again, regarding the use of large percentages of magnetic concentrates, 35 years ago nearly or quite as fine ores were used up to 100 per cent. Mr. Bachman has used that percentage at Port Henry, and I understand has not had a slip in several years, but he uses a charging-apparatus that distributes some of the charge to the center, designed as his experience has dictated. Mr. Langdon also used as high as 90 per cent. of concentrates, I believe of greater fineness than are turned out there now, with a mixed coke-and-anthracite fuel charge, also using a center charging-apparatus. The Bay State furnaces used Cheever fine ore—a magnetite—which averaged as fine as wheat, with furnace lump anthracite coal, first as open top, hand-filled furnaces, and about 1866 they were the pioneer bell-and-hopper-charged furnaces in the United States. Their record was an enviable one, as to both fuel-economy and regularity. In each case the charging was well to the center. In the hand-filling period, it was pretty evenly distributed over a

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rather narrow top, and when filled by a bell, Mr. Foote used one of 7.5 ft. diameter on a 14-ft. stock-line, which, at the present day, would be considered a small bell for so large a top. The bosh was 16 ft. and the height 66 ft.

I can cite two cases of explosions which seem to require still other explanations—a coke-furnace which slipped while the wind was off, and had been off for some time, accompanied with considerable damage and great loss of life; another—an anthracite furnace—which exploded 10 hr. after lighting, and at least 20 hr. before the wind would naturally have been turned on, according to the practice at that date, lifting out the hopper and breaking both bell and hopper, besides cracking the lining down 30 ft., opening it in four places from 3 to 4 in. at the top. The real cause of the damage was neglecting to chock the bell open, and when the explosions came, the bell closed like a huge check-valve, and, as all down-comer outlets were closed, something had to yield.

The effect of moisture in the blast on the working of the blast-furnace seems to have been well understood by the oldest iron-masters. In Truran's *Manufacture of Iron*, written before Neilson's invention of the hot blast, he says, as a result of 20 years' observation of the workings of 19 blast-furnaces at Dowlais, Wales, that "there was a difference of about 16 per cent. (I speak from memory) in favor of the cold months."

Yet Mr. Robinson finds a difference of only 21 lb., or about 1 per cent., in fuel-consumption for a period of ten years at the South Chicago furnaces. It is worthy of note that the Dowlais furnaces were cold-blast at that time, and Truran's observations must have begun nearly 100 years ago.

Until Mr. Richards and Mr. Johnson made their very exhaustive calculations which satisfactorily account for the saving in fuel, over and above that directly due to having less water to decompose, we were obliged to class "dry" blast with "hot" blast, the effect of which cannot be accounted for by actual calories carried in alone. In each case the first result is to concentrate the heat at the tuyere, where it belongs, which in turn sets up a secondary set of reactions, or modifies the usual ones. While it is true that a blast-temperature of 1,100° or 1,200° F. would have brought the fuel-consumption down to 1,700 lb. or thereabouts, it would seem that to cite that fact is

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begging the question, as that expedient was and is open to Mr. Gayley. Supposing that circumstances had permitted Mr. Gayley to have applied dry-blast to a furnace using blast up to the highest practicable limit of heating, say from 1,800° to 2,000° F., and that the same saving had been attained over the fuel-consumption to be expected at such temperature of blast, what possible theory could have been set up to deprive dry-blast of the whole credit? The facts, as they appear at the present time, are that dry-blast was applied, and the net result was that 400 lb. of fuel was saved; a greatly increased product and greater regularity of working attained; and I believe that Mr. Gayley and "dry-blast" should receive the whole credit, no matter by what roundabout way such favorable results were obtained. It is probable that Mr. Gayley expected about what he would attain by turning summer conditions into winter ones—as he certainly would not have felt justified in spending so much money in order to save 3 or 4 per cent. in fuel-consumption, due to the less amount of water to be decomposed, and I believe he had it all carefully figured out in advance.

Mr. Johnson's "critical heat" theory seems to open up a new field for metallurgical calculations, and to explain satisfactorily many interesting and important furnace-reactions in a new and simple way. This new theory seems to amount to the self-evident proposition that that furnace which is the hottest has the greatest capacity for work in its line. Whatever the temperature may be in any given case, it depends upon four factors—viz., carbon-combustion to CO, blast-temperature, humidity, and temperature of descending materials (the latter being a resultant of the other three factors, two of which are controllable within limits—blast-temperature and moisture). So, going a step further, it amounts to saying that the furnace with the most powerful stoves, and the most efficient refrigerating-apparatus, is best prepared to fulfill its office. The degree of oxidation of the top-gases may also have some effect on the temperature of materials *en route* for the tuyere region. If Mr. Johnson could have gone enough further with his calculations to have fixed a necessary amount of heat for superheating iron and slag, and doing the other things left by him for "available heat" to do, it would have been clearer to me. Now, if that

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had been done or can be done, then it would appear that there would have been no "H_a" or remnant left, with economical practice, or if any, it would mark the unnecessary or surplus heat—the lack of full economy. It might also be considered as a "heat-reserve," measuring the difference between actual economy and the "dead-line," so to speak. In making calculations to determine how much slag can be made and fused per ton of carbon, I have assumed that the available heat is what is left after deducting from the total heat developed, the blast-furnace fixed charges due to that carried off by gases, radiation, cooling-appliances, expulsion of water from materials, decomposition of water in blast, etc.; though that is a different proposition, as it deals with total heat-units developed, while Mr. Johnson deals with temperature, as I understand, though I confess that I do not understand it very well. I hope and expect that a paper of such character will bring out a full and clear elucidation of so important a theory, and if it proves to be sound, Mr. Johnson will be entitled to many thanks for providing us with such a handy slide-rule for blast-furnace calculation.

Mr. Johnson's "critical heat" appears to be comparable to the "clinkering-temperature" of a rotary cement-kiln—a temperature at which chemical combination has taken place, below that of complete liquid fusion. Mr. Hilbig, of the German-American Cement Co., La Salle, Ill., places that temperature between 1,300° and 1,400°C.; taking it at 1,350°C. (or 2,462°F.), it agrees with the 2,750° given by Mr. Johnson in the case cited within 288° F., which might be sufficient to bring the materials up to "what may be conveniently called the free-running temperature of the cinder," which Mr. Johnson designates as the "critical temperature," and thus would seem to sustain the theory.

Mr. Bell was of the opinion that hydrogen had but little, if any, reducing action in the iron blast-furnace. By taking special precautions he always found it in the top-gases, and gave examples showing the amount to be from 0.84 to 1.01 per cent., calculated on the CO, or roughly, say 0.21 and 0.23 per cent. on the total gases—an amount so insignificant as to be readily lost in an ordinary gas-analysis. However, hydrogen can be qualitatively determined in the top-gases, beyond the shadow

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of a doubt, without resorting to gas-analysis. In steaming a too-hot furnace to control silicon and to make "smoky" or "gray" gas burn, it was noted that a $\frac{3}{8}$-in. jet of steam at 30 lb. pressure injected into a tuyere, would show the hydrogen flame at stoves and boilers in about one minute.

Reducing the steam-pressure to the blast-pressure made the volume of steam about 1 per cent. of the blast-volume. Injecting steam for the purpose named is a very valuable expedient, and as it is required just when the furnace is too hot, there are no bad effects to fear. In case of gas blowing-engines "steaming" should be a remedy for smoky or incombustible gas. Usually steaming need not be long maintained.

Again, sometimes the only visible symptom of a leaky tuyere or cooling-device is the appearance of hydrogen in the burning gas, generally accompanied by an abnormal amount of steam.

I recall an instance where the burning gas was seen from a distance of about 3 miles issuing from the boiler and stove stacks, which were respectively 115 and 165 ft. high. Upon arriving at the furnace no signs of leaking were visible, but the leak was finally found to be from a cooling-plate about 5 ft. above the tuyeres—a $\frac{3}{8}$-in. hole and 30-ft. head of water.

Regarding the hygrometer asked for by Mr. Johnson, I would say that when I left for Mexico, 18 years ago, I had practically finished a large recording-instrument to diagram 14 different things pertaining to blast-furnace practice. Among the records to be made were the following:

Barometric pressure, air-temperature, air-humidity, revolutions of engines, pressure of blast, blast-pressure divided by blast-volume (giving resistance to passage of blast through the furnace for 1,000 cu. ft. of blast, and kept as $\frac{P}{R} = r$), temperature of blast, steam-pressure, vacuum, time lost, number of charges, time of charging, weight of elements of charges. A margin was also provided to fill in by hand the quality and grade of iron and the analysis, the whole card to be a complete furnace-journal. The recording and direct-reading hygrometer, which I may explain later, is about the simplest of the lot. The pyrometer and the instrument for dividing blast-pressure by blast-volume required the most study, but finally

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were resolved into quite simple machines. A record was kept of $\frac{P}{R} = r$ for three and a half years, and it should afford some consolation to anthracite furnace-men. It showed that when two-thirds anthracite and one-third coke was used, the resistance per 1,000 cu. ft. of blast was 0.55 lb., and that when all anthracite was used the resistance was 0.77 lb. per 1,000 cu. ft.; or that it required 40 per cent. more power to blow an anthracite furnace than a one-third coke one. So when each makes its own steam, without firing boilers, the coke-anthracite furnace has more heat available for making iron. When the furnace was working normally the resistance-record was a straight line, regardless of pressure or volume, showing that the resistance was directly as the volume. When the resistance line "r" ascended or descended from the normal, it generally indicated a hotter or a cooler furnace, respectively.

The use of caustic lime, according to Bell, does not seem to promise much, if any, advantage, for the reason that at a temperature below that at which limestone gives up its CO_2, caustic lime has a strong affinity for CO_2, which would have to be again expelled lower down. I can see two other possible, though minor, disadvantages in the use of burned lime. I have had occasion to examine many samples of what was supposed to be caustic lime, but all had CO_2 in considerable quantity—from 10 to 20 per cent. and more—enough to be appreciable in fluxing where lime has to be carried to the safe limit, and probably much more burned lime would be blown out of the furnace than if raw stone were used, which would also tend to upset the fluxing-calculations.

As to the practical limit of the hot-blast temperature, I was in a position to maintain an average of 1,392° F. for more than two years, and found it a very comfortable temperature to carry, and not at all destructive to stoves or connections, providing they did not leak. Heat up to 1,700° F. was occasionally carried, and sometimes even more. For such heats the copper ball of a Siemens pyrometer was not durable, and one would show an error of 300° in a week. So platinum was substituted. For another purpose than iron-making I have just designed stoves to give from 1,000° to 1,100° C., which may be about the limit. Such high heats will require that all connections

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be lined, even the belly-pipes, and also it will be necessary to wash the gas; otherwise the flue-dirt and fume will fuse and cut the walls of the combustion-chambers, and likely permanently seal up the checker-work at the top.

In the original fire-brick stoves at the Cedar Point furnace, at 1,400° F., no dust left the combustion-chambers, but all fused into cinder, and occasionally had to be dug out. A lining in belly-pipes and first elbow of tuyere-connections will cause some trouble in cleaning them, should they ever fill with slag, in which case it would be best to have spare ones at hand and clean the others at leisure. The matter of relining need not require more than from 10 to 15 minutes per tuyere, and the cost would be inconsiderable.

Supplementary Note.*

In quoting Mr. Truran on the subject of moisture in the blast, I was obliged to rely upon my memory of what I had read 40 years ago, and I made some mistakes, to which Mr. Frank Firmstone has kindly called my attention.

I have just re-read Truran's *The Manufacture of Iron*, 2d edition (1867), and now write with more accuracy. On page 112 Mr. Truran says: "Taking the average of five years, selected promiscuously from twenty-two years' working, we find that at the foundry iron furnace the yield of coal per ton of pig-iron was, in the winter months, 49.7 cwts.; spring, 52.2 cwts.; summer, 53.1 cwts.; and autumn, 55.4 cwts. The excess of autumn over the winter months, 5.7 cwts., is equal to an increase of 11 per cent. At the forge iron furnace the yields in the winter months are 43.6 cwts.; spring, 44.2; summer, 44.6; and autumn, 45.8 cwts. The excess of autumn over winter, 2.2 cwts., is equal to 5 per cent. The variation of yield with the season is still more marked with the ballast iron furnace, the yields being, in winter, 43.2 cwts.; spring, 44.1; summer, 50.1; and autumn, 49.5, or 6.3 cwts. more in autumn than in winter, equal to 13 per cent. nearly." Recasting these data, the difference in fuel-consumption in favor of the dry period of the year is:

	Per Ton of Iron. Pounds.
Foundry iron furnace,	638
Forge iron furnace,	246
Ballast iron furnace,	705

* Received July 26, 1907.

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I can find no date or data that will give even a close approximation of the time that Mr. Truran wrote his work. It appears to have been written somewhere between 1827 and 1854, though he goes back to 1797 for some data. I can only say that it was not "before Neilson's invention of the hot blast," as I stated in my paper, for "hot blast" was discussed perhaps more thoroughly than any other subject.

Truran discussed "hot blast" on the same lines that "dry blast" has been recently considered by some writers—viz., from the stand-point of heat-units alone—those carried in by the hot blast on the one hand and, on the other, those not needed to decompose the lesser quantity of water entering the furnace. In both cases the main point—the total effect—was missed.

Notes on the Gayley Dry-Air Blast-Process.

Discussion of the Paper of C. A. Meissner, *Trans.*, xxxvii., 201 to 216.

J. E. Johnson, Jr., Glen Wilton, Va. (communication to the Secretary*):—Mr. Meissner announces early in his paper that one of its purposes is the discussion of my paper entitled, Notes on the Physical Action of the Blast-Furnace,[1] with which he disagrees in part; and as the second stage of the discussion of this important subject has centered around his paper, some reply by me may not be amiss.

Unfortunately, Mr. Meissner has not indicated a single statement or conclusion in my paper that he wishes to controvert; but as there is evidently a difference of opinion between us I will disregard this fact, and will discuss those conclusions of his that seem to me not to be justified by the facts of general practice, or even by the data which he produces to support them.

Let me say, however, that I am a believer in the dry-air

* Received July 3, 1906, too late to be printed for the London meeting; subsequently revised in minor details by the author, and returned for publication Nov. 19, 1906.

[1] *Trans.*, xxxvi., 454 to 488 (1906).

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blast; in fact, a more consistent one, apparently, than Mr. Meissner, since to me the great advantage of the dry-air blast appears to be its dryness. In my opinion, it is not necessary to produce factitious explanations for the success of this great improvement.

Mr. Meissner's first contention is that natural-air blast has not shown the increase in output and the decrease in fuel-consumption when the air is cold and dry in winter, which was caused by cold, dry blast artificially produced. He cites, in support of his contention, the records of 90 furnaces, extending over a period of from four to six years, which failed to show these results.

I shall deal presently with the explanation of this fact, but first I wish to say that this contention is opposed to the experience of almost every practical furnace-man of my acquaintance, excepting some of those using Lake Superior ores.

Mr. Gayley himself, in the specification for his original patent on drying the blast by refrigeration, says: "Thus in the summer months, because of the greater content of moisture in the air, 200 lb. more fuel on an average are required to produce a ton of pig-iron than in the winter months." Mr. Gayley was at that time managing the largest blast-furnace plant in the world; and better testimony on such a subject it is impossible to imagine.

About 8 or 9 years ago I took the records of fuel-consumption and output of a furnace in Virginia for three years, week by week, taking the average for corresponding weeks, and omitting from the record those weeks in which the furnace was shut down for repairs. The furnace was run during the entire period on the one kind of ore, with a uniform blast-temperature (850° F.), and made the one kind of iron (basic). In order not to show the absolute coke-consumption, and to show the variations more plainly, the quantity in excess of a certain fixed amount was plotted, as shown in Fig. 1. The results hold good to the present time; that is, the same annual variations of fuel-consumption and output occur, except as they may be masked by periods of better or worse working, which, unfortunately, are generally beyond the furnace-man's control. Throughout the South, wherever close attention is paid to fuel-consumption, the same results hold to a greater or less degree,

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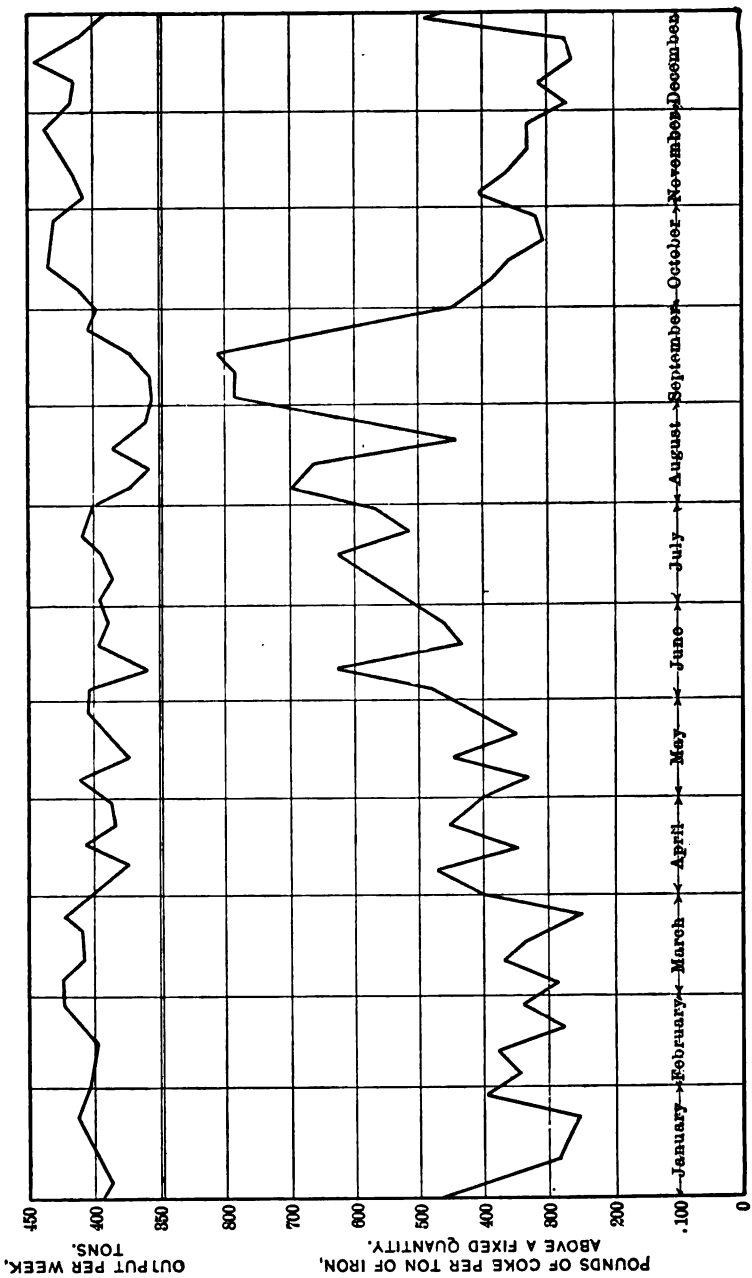

FIG. 1.—WEEKLY AVERAGE OF COKE-CONSUMPTION AND OUTPUT FOR THREE YEARS OF A VIRGINIA BLAST-FURNACE.

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as far as my experience extends, and that also of others with whom I have discussed the subject. In fact, this condition has so long been one of the banes of furnace-men's existence that further discussion of the point is hardly needed. The effects are more definitely measurable in the case illustrated by Fig. 1 than in most others, for the reason that the percentage of variation in available heat caused by a given amount of moisture is much greater with a low blast-temperature than with a high one. For instance, from the diagrams published in my paper above referred to, the available heat with 0.25 lb. of moisture and 1,250° of blast-temperature is 1,910 B.t.u.; for 1 lb. of moisture at the same blast temperature, it is 1,650 B.t.u., a reduction of 14 per cent.; for 750° blast-temperature, with the same amount of moisture, the available heats are 1,300 and 1,045 B.t.u., a reduction of 20 per cent., or about 45 per cent. more than in the previous case.

In this connection another point is to be borne in mind. When the furnace is carrying a good burden the gas is lean, and of relatively small quantity; consequently there is relatively little left for heating the stoves, after the boilers have been supplied, and the blast-temperature is, therefore, comparatively low. When, on the other hand, the burden is low, the gas is abundant and rich, and there is consequently a larger supply for the stoves and the blast-temperature can be suitably raised. The result is that the work of the furnace tends to be equalized throughout the year; that is to say, when the furnace might be able to carry a good burden, it can only go as far in that direction as the reduction of blast-temperature (due to the impoverishment of the gas) will permit. If, on the other hand, the furnace tends to work badly, the increased heat in the blast obtainable from the enrichment of the gas tends strongly to correct this condition.

In the furnace, the operation of which is shown by Fig. 1, this was not the case, since it was equipped with iron-pipe stoves, on which the maximum permissible temperature was 850°; and this was always maintained, even though firing the boilers with coal became necessary in consequence of a deficient gas-supply.

This reasoning does not fully apply to the 90 furnaces of which Mr. Meissner speaks, because many of them do not use

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all the heat that their stoves would permit, for reasons well known to furnace-men—namely, (1) the liability to make the furnace "stick," and (2) the desire to have in reserve enough heat to help the furnace through a sudden "spell." But in this case a new factor becomes operative; this is, that the furnace-man, who fears to stick his furnace with too high a blast-temperature, will always use all the heat that the furnace seems willing to take. Consequently, when the summer months come and the humidity rises, thus producing a chilling action in the hearth, these furnace-men naturally will order the stove-tender to carry a higher blast-temperature, because they see that the furnace will stand it.

These considerations explain to a very great extent, if not wholly, the reason why furnaces, especially those using Lake Superior ores, with their liability to stick, do not show the difference in fuel-consumption between summer and winter conditions which might be expected. Add to this the fear of every furnace-man that his furnace will be caught by a slip, or a dose of water, or a violent change in the humidity, at a time when it is already carrying all the burden under which it can stagger, and his desire to have a reserve with which to meet this possible change, and the conditions described by Mr. Meissner are completely explained.

It is perfectly well known to most furnace-men that a furnace will work more regularly, with less liability to slips and other derangements, on a heavy burden than on a light one. Therefore, when the furnace-man feels himself secure against atmospheric changes, he is justified on that account in carrying all the burden he can (as we have Mr. Gayley's own word that he did). The effect of this increase of burden is to further steady the furnace, and reduce the irregularities other than those caused by variations in the humidity. This makes the effect of the dry-blast cumulative in two or three directions.

In this connection, it may be well to state that owing to regularity of work at the Virginia furnace, the records of which are graphically shown above, it was the custom to put on all the burden that the furnace could carry, the danger from slips and other accidental derangements being a minimum.

It has been generally declared that the diurnal variations of the humidity are very great, and that the removal of these vari-

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ations is in itself a great advantage, but I confess that both the facts and the results seem to me to oppose this view. There is a diurnal variation in the humidity, and on some occasions it is rapid and of great extent; but these occasions are comparatively rare in most seasons; and many years of patient but unsuccessful endeavor to make the facts fit the theory, and to prove that a furnace works better at night than during daytime in summer, have convinced me that the diurnal variations are, under ordinary circumstances, no more than a furnace in good condition can meet with its reserve of heat, without showing ill-effects.

On pages 204 and 205, Mr. Meissner has made some comparison of actual results with those indicated by my diagrams. Unfortunately, a serious typographical error has occurred, for which Mr. Meissner is not responsible, but the effect of which is lamentable, as it tends to throw complete discredit on the diagram. The entire sentence is as follows: "According to Mr. Johnson's diagram, this increase from 850° F. to 1,200° F., on a blast containing 2 grains of moisture per cu. ft., should increase the 'available heat' from 1,450 to about 1,850 B.t.u. Theoretically, this would reduce the coke-consumption of 1,875 lb. for the former temperature, to about 1,270 lb. for the latter." The words "about 1,270 lb." should be "about 1,470 lb.," a result which is, of course, very low, but not so far from the very best practice as to be ridiculous.

Through the kindness of Mr. David Baker, I obtained a brief statement of the work of No. 6 furnace, at South Chicago, for 1897, which shows an average fuel-consumption of 1,580 lb., with a blast-temperature of 1,130°. Considering that this covers the entire year and that the average humidity was, therefore, not far from 0.5 lb. of moisture per 1,000 cu. ft. of air during the entire time, the available heat for this case by my diagram is about 1,690 B.t.u. This corresponds, on the basis given by Mr. Meissner, to 1,620 lb. of fuel, more than was actually used, so that if the blast were dry and the temperature raised to 1,200°, it is not unreasonable to suppose that on a similar ore-mixture a fuel-consumption of very nearly 1,470 lb. could be reached, as the diagram indicates. An additional confirmation of these figures is to be found in the actual work of the Dover furnace, under the management of Mr.

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Arnold K. Reese, as described by him,[2] in which the fuel-consumption was about 1,660 lb. and the blast-temperature about 1,150°. I think it may be claimed, therefore, that the results taken from my diagram show as close an agreement with the actual results of blast-furnace operation as would be expected by those who have had actual experience with that uncertain apparatus. Mr. Meissner's own figures prove this clearly for the lower blast-temperatures, and, when properly corrected, prove it as well as can be expected for the higher ones, also.

Mr. Meissner's Argument for Perfect Uniformity.—The second portion of Mr. Meissner's argument is devoted to proving that the advantage of the dry-blast lies principally, not in its dryness, but in its uniform density and content of oxygen. On this latter subject, much has been said by European metallurgists; and it seems worth while to point out that the variation in oxygen-content of air with the amount of moisture present is not rapid, and that it actually increases as the moisture increases after a certain point is reached. To show this, I have reproduced in Table I. Mr. Meissner's Table II., and have extended it to 112° F., and have added on the right a column showing the oxygen in pounds per cubic foot of saturated air at the different temperatures.

TABLE I.—*C. A. Meissner's Table II., with Oxygen per Cubic Foot of Saturated Air Added.*

Difference in Saturated and Dry Air per cubic foot at different temperatures.

Grains of Moisture at 100 Per Cent. Saturation.	At	Saturated.	Dry	Oxygen per cu. ft. Pounds.
0.55	0° F.	equals 0.0863 lb. per cu. ft. of air,	or 0.0864	0.01990
0.91	12° F.	equals 0.0841 lb. per cu. ft. of air,	or 0.0842	0.01943
2.12	32° F.	equals 0.0805 lb. per cu. ft. of air,	or 0.0807	0.01870
4.38	52° F.	equals 0.0772 lb. per cu. ft. of air,	or 0.0776	0.01819
8.54	72° F.	equals 0.0739 lb. per cu. ft. of air,	or 0.0747	0.01780
15.75	92° F.	equals 0.0707 lb. per cu. ft. of air,	or 0.0720	0.01772
27.68	112° F.	equals 0.0631 lb. per cu. ft. of air,	or 0.0694	0.01800

It will be seen that this reaches a minimum at about 95° F., and thereafter increases, for the reason that air contains only 25 per cent., by weight, of oxygen, while water-vapor contains 89 per cent. It may be objected that the oxygen is combined with hydrogen, and therefore does not count; but this fact does not

[2] *Trans.*, xxvii., 477 to 485 (1897).

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interfere in the least with its combination with the fuel to form CO, and the hydrogen released at the same time is a desirable reducing agent, so that if the question of heat be ignored, as it has been by so many in this discussion, the blast-furnace should really work better with saturated air above 100° than it does with dry-air at lower temperatures. No one has actually claimed this, but it is a not altogether unreasonable deduction from the arguments of many.

Referring to Mr. Meissner's attempt to trace the results obtained to uniformity of temperature of the dry-blast, on page 209 *et seq.*, I think that they may all be dismissed with the verdict "Not proven." The variations in the humidity as shown by Mr. Meissner's comparative tables are, it is true, greater than the variations in temperature of the dry-blast; but while the relative variation of humidity is considerable, the absolute variations are negligible from a metallurgical point of view. The diagrams given in my paper show a greater variation in probable fuel-consumption for a given variation in moisture, by from three to five times, than any other method of calculating this loss; therefore, if the variation in humidity shown by Mr. Meissner's tables were of importance, this would be shown by the corresponding variation in available heat as taken from my diagram. The extreme variation in moisture given in Table V., with the exception of one month, is 1.25 grains, which would correspond to a variation in available heat of about 60 B.t.u. out of a total of 1,500 B.t.u., or about 4 per cent. On any other basis of calculation than mine the variation would be less than 1 per cent. To claim that the superior work of one period over another was due to any further reduction in this small quantity is utterly inadmissible; in fact, Mr. Meissner himself gives the clew to better work of No. 3 furnace, than of No. 1 on dry-blast, when he speaks of the "troubles at the furnace due to accidents and repairs and to leakage of bosh-plates." To attempt to compare the work of two furnaces, of which one had these troubles and the other was free from them, in order to evaluate a gain in uniformity of dry-blast temperature, is hardly less than preposterous. In fact, the differences as between variations of humidity and variations in dry-blast temperature given by Mr. Meissner's table are so utterly insignificant in comparison with the variations of furnace-work under the most uniform and

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favorable conditions that any inferences so drawn are of no more value than an attempt to determine the distance of the Pole Star by taking its parallax from opposite ends of a two-foot rule. Above all, to give these comparisons without giving in detail the variations in hot-blast temperature which accompanied them, and which are almost certain, in some cases, to have been of many times greater moment, is utterly useless.

On page 214, Mr. Meissner claims that the calculated effect of higher blast-temperature upon fuel-economy is not realized in general practice. From this statement I am forced to dissent. So far as known to me, there has seldom been a blast-furnace run remarkable for low fuel-consumption under given circumstances without a high blast-temperature; and while a high blast-temperature may not always give a low fuel-consumption, as compared with practice on Lake ores, high blast-temperature and furnace-management has, in almost all cases, given a lower fuel-consumption than a lower blast-temperature under the same conditions. The exception to this statement is the one with which Mr. Meissner is familiar—namely, that of Lake ore practice, where limitations caused by liability to stick and bad furnace-work prevent obtaining from high blast-temperature those benefits which are almost universal under other conditions.

Mr. Meissner says that an increase of 300° in blast-temperature would probably not result in a decreased fuel-consumption of 400 lb. per ton of iron, and that practice indicates 50 lb. of fuel saved per 100° increase in blast-temperature. I have dealt with this probability in the early part of this discussion; but it may be well to point out again the fact on which special emphasis was laid in my paper—namely, that the fuel-economy of the blast-furnace is subject to a double limitation: first, that arising from the total amount of heat generated per pound of iron produced; and second, that arising from the fact that a sufficient portion of this heat may not be available above the critical temperature to complete the smelting-process. One of these depends, roughly speaking, on the degree of oxidation of the carbon in the waste gases; the other depends upon the temperature and the dryness of the hot-blast. These two limitations are practically quite independent of one another; and whichever of them comes into action first will limit the econ-

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omy of the furnace, even though the other still have an ample margin. This is a theorem which I believe to be absolutely essential to the proper understanding of the operation of the blast-furnace. Without it, many unwarranted conclusions will be drawn, and many points will fail to be understood. Its bearing on the present matter is, that if, in a given furnace, the top gases are already oxidized to the maximum possible extent for the given conditions, further increase in the temperature of the blast will have practically no effect. If, on the other hand, the gases are much less oxidized than is possible under the given conditions, then increase in blast-temperature will be followed by economy of fuel, in all those cases in which the increase in temperature does not occasion a tendency to stick and consequent irregularities. The conditions controlling this last occurrence need not be discussed at length here, and are hardly germane to the present discussion.

As to the reduction in the power required, there is no doubt that this is a factor worthy of consideration, but one of the reasons for it has been entirely overlooked. This is, that the furnace requires less blast to burn the smaller quantity of coke charged, and as the pressure to drive the blast through the furnace varies nearly as the square of the volume used, the pressure on the furnace falls materially, and therefore the work of the blowing-engine is very considerably reduced, altogether apart from the saving made by the reduction in the volume of the air, due to its lower temperature at entrance into the blowing-cylinders. That this saving also exists is not to be denied; but considered by itself, it would be a very expensive one at the price paid for it—namely, the power required to refrigerate the blast.

In the discussion by Prof. Richards on this subject at the Washington Meeting, May, 1905, and at the Bethlehem Meeting, February, 1906, he has failed to grasp the importance of the conception of a critical temperature, and of the double limitation to fuel-economy just mentioned. As a consequence, his analysis, while interesting, does not advance our knowledge of the subject; since he attempts to evaluate causes to which it is not possible to attach exact values, and the introduction of which is unnecessary to account for the results achieved. Prof. Richards's heat-balance, in his paper of May, 1905, has its great-

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est interest in showing, with a little additional calculation, a fact which he ignored—namely, that the total quantities of heat actually used in the furnace per pound of iron made were identical (within the limits of such calculations) with and without the dry-blast, thereby disproving the very foundation of many arguments concerning this process.

To Prof. Howe I am under lasting obligation for having restated with his faultless clearness of expression the views set forth in my paper, and for having illustrated them at such length, and with such clearness as to render their acceptance much more certain and more rapid. I think, however, the fact has been generally overlooked that in my paper I not only called attention to the existence of a critical temperature and to its importance, but also was at some pains to determine what this critical temperature was, an effort in which I expended more than a year. It should not, therefore, be thought that the existence of a critical temperature depends upon mathematical demonstation or that it may be fixed by mathematical analysis, because, if my views are correct, the critical temperature is a metallurgical fact which is determined within a comparatively narrow range by the circumstances of any given case.

In conclusion, it is probably safe to repeat that the great advantage of the dry-blast is its dryness, and that that portion of its results which cannot be explained directly on that account alone is completely accounted for by the moral and material secondary advantages which come from it, as explained above.

C. A. MEISSNER, New York, N. Y.*:—I have listened to the reading of Mr. Johnson's paper very carefully, and I can so far only say that I have nothing whatever to change in the notes that I made at the time. I think that a careful study and analysis of the wording and meaning of what I then said will practically answer most if not all of Mr. Johnson's contentions. I have given in that paper purely a statement of facts; I have not theorized, but have given the facts as I found them after more than two years' study of both dry-blast and normal blast, and to go into any detailed discussion of Mr. Johnson's paper would only be taking up your time, be-

* Oral discussion at the New York Meeting, April, 1907.

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cause, as I say, I cannot change any of the facts I have presented. I think the deductions that the facts have shown are good. I have gone over them again recently and I have nothing whatever to change in them at this time.

HENRY M. HOWE, New York, N. Y.*:—To say that the reason why dry-blast causes the remarkable fuel-saving which has been shown is that it leads to regularity, seems to me absolutely no explanation at all, because it leaves us wholly in the dark as to why regularity should cause so great a saving of fuel. It simply pushes the question one step further off, throwing absolutely no light on the question of the true cause of the saving of fuel.

On the other hand, the explanation that the dry-blast widens the margin between the temperature developed and the critical temperature of the blast-furnace operation, as put forth by Mr. J. E. Johnson, explains the saving readily, clearly and convincingly, at least if we couple with it the necessary amendment which I have offered.[3]

MR. JOHNSON:*—I wish to say that, contrary to Mr. Meissner's impression, I did study his paper and conclusions very carefully before I wrote my own. It is only fair to say that he had the facts, whereas I had only the observation of allied facts, and I was exceedingly careful to give due weight to that evidence; but the result of all my thought on the subject is to confirm my original conclusion in spite of the very strong case Mr. Meissner has prepared, and in my paper those interested will find a reason why there is a lack of variation between dry-blast and normal blast in the summer- and winter-conditions in the Pittsburg region, which is too technical to go into here.

* Oral discussion at the New York Meeting, April, 1907.

[3] *Iron, Steel and Other Alloys*, by H. M. Howe, 2d edition, pp. 457 to 475.

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The Constitution of Mattes Produced in Copper-Smelting.

Discussion of the Paper by Allan Gibb and R. C. Philp, *Trans.*, xxxvi., 665.

ALLAN GIBB, Queensland, Australia (communication to the Secretary*):—It is gratifying that Mr. Edward Keller,[1] who has done so much work elucidating the principles of copper-metallurgy, should have subjected only that portion of our investigation relating to magnetic iron oxide to anything like adverse criticism.

Situated as we were on a mining-camp, it was impossible to undertake work that would cover all the intricate possibilities connected with the materials commonly classed as matte. We therefore began the investigation on matte produced from ores that contained no other components than the sulphides of copper and iron that would be likely to enter into the composition of matte. We were, accordingly, free from any possibility of magnetic iron oxide entering the matte, unless it were formed in the operation itself. The work done on this subject was incomplete, and the results thereof recorded as giving only negative evidence as to the absence of magnetic iron oxide. It would have been better to have left the matter open, as "not proven." The trend of the investigation was to account for the insufficiency of sulphur in matte to satisfy the requirements of ferrous and cuprous sulphides. So far as our deductions are correct, it was unnecessary to consider the (in our samples) somewhat doubtful constituent, magnetic iron oxide.

The mattes that were investigated were among those mentioned by Mr. Keller, "which scarcely show any magnetic property." The samples showed no such property. There was no unaccountable variation in the specific gravity, which only varied, as would be expected, with the proportion of ferrous sulphide. I have had experience in smelting ores of which the gangue was a siliceous magnetite, both in blast- and reverberatory furnace. The mattes so produced contained magnetic

* Received Aug. 4, 1906.

[1] *Trans.*, xxxvi., 837 (1906).

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iron oxide, sometimes as much as 7 per cent. of iron being present in this form. On treating such mattes with oxidizing solvents, the proportion of magnetic iron oxide varied somewhat with the solvent used, but a fixed proportion was obtained when concentrated sulphuric acid was mixed with the solvent. This residue could be dissolved only after fusion with fusion-mixture. Upon resmelting these mattes in blast-furnaces, in which the volume of blast was largely in excess of that required to oxidize the carbonaceous fuel, the proportion of insoluble magnetic iron oxide was reduced. Apparently, even under highly-oxidizing furnace-conditions, magnetic iron oxide is decomposed, presumably by silica and sulphides. I have seen magnetic iron oxide crystals produced in a furnace smelting erratically, but I am of the opinion that normal work, whether reducing or oxidizing, does not tend to form this compound.

I do not grant to "*das chemische Gefühl*," nor to chemical precedent, the all-embracing infallibility apparently expected by Mr. Keller. Chemical investigation, exactly as with metallurgical, has for its object the elucidation of certain phenomena under certain conditions, and there are innumerable chemical reactions that are modified or entirely reversed by varying the conditions. A case somewhat similar to that to which Mr. Keller takes objection caused me trouble some years ago. According to records, metallic iron will precipitate antimony from solution without having the same effect upon tin. This difference constitutes a rough analytical separation for the two metals. Under certain conditions of saturation this separation breaks down, tin being precipitated almost as readily as antimony. Returning to the reaction under criticism, I have obtained products from the direct fusion of pure sulphides of copper and iron in crucibles that do and do not give residues of magnetic iron oxide when treated with nitric acid and potassium chlorate. As regards the solubility of the magnetic oxides of iron assumed to be precipitated by the oxidizing solvent, it is not uncommon that an ignited precipitate is quite unacted upon by solvents that readily dissolve the precipitate before ignition. This may, as suggested by Mr. Keller, be due to change in molecular segregation.

Mr. Keller appears to have given greater prominence to this

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subject than we intended for it. That magnetic iron oxide may be held, either chemically or physically, by mattes is undoubted, but its presence is not explanatory of the fact that the proportion of sulphur in mattes is less than that required to satisfy the requirements of the copper and iron. Further, I consider that when complex mattes are treated with oxidizing solvents, part of the iron may, under conditions at present unknown to me, be converted into magnetic iron oxide.

Referring to the magnetic properties of solid solutions of iron in ferrous sulphide, the saturated solutions are strongly magnetic, whereas as much as 15 per cent. of iron may be added to ferrous sulphide before the solid solution becomes feebly magnetic.

Mr. Philp is at present in London, and I have had no opportunity of discussing the criticism with him. The foregoing will, accordingly, represent my personal views of the matter.

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Comparison of American and Foreign Rail-Specifications, with a Proposed Standard Specification to Cover American Rails Rolled for Export.*

A reply to the Discussion of the Paper of Albert Ladd Colby, *Trans.*, xxxvii., 900.

ALBERT LADD COLBY, New York, N. Y. (communication to the Secretary†):—I observed (*Trans.*, xxxvii., 585) that to obtain tenders from several American mills, the foreign engineer should modify his maximum phosphorus to 0.10 per cent., not as Mr. E. Windsor Richards quotes me, that no American rail-maker can produce rails under 0.10 per cent. of phosphorus. Mr. Richards claims that no English engineer would agree to accept 0.10 per cent. of phosphorus on any terms whatever. Mr. Webster writes that only after an engineer has been convinced that rails can be made in America equal in every respect to those he is getting abroad, should he be asked to modify any conditions in his specifications.

I know it to be a fact that during recent years in many cases, as, for instance, when delivery could be secured more promptly in the United States than in Europe, the consulting engineers of several British and colonial railroads have frequently agreed to modify their phosphorus requirement to a maximum of 0.10 per cent., and the rails delivered[1] under these modified specifications have, under like conditions, given just as satisfactory service and proved to be no more brittle than similar sections of 0.07 per cent. phosphorus rails, rolled in England under the same specification except without modification as to phosphorus.

Furthermore, it would appear from the analyses of British rails, quoted by E. A. Dancaster,[2] in his discussion of Mr.

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* *Trans.*, xxxvii., 576 to 627 (1907).

† Received Feb. 13, 1907.

[1] *Export Statistics*, p. 638.

[2] *Journal of the Iron and Steel Institute*, vol. lxviii., pp. 343 and 345 (1905).

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Thomas Andrews's paper on the Wear of Steel Rails on Bridges, that the limits of 0.06 per cent. in the older British specifications, and of 0.07 per cent. of phosphorus in the British Standards Committee specifications, quoted by Mr. Richards, are not lived up to by home mills.

I recommended a maximum of 0.10 per cent. of phosphorus in my suggested specification, which I have defined as a contract to be strictly lived up to, because it insures competition and because there is no evidence that the large tonnage of low-carbon American rails which have been exported have, because of a maximum content of 0.10 of phosphorus, instead of 0.07 or 0.08 per cent., failed or proved brittle after service. I freely admitted that phosphorus was the most undesirable constituent of steel, but, as Mr. Stead and Mr. York have pointed out, rail-failures are frequently wrongly attributed to the phosphorus-content, and, on the other hand, there is abundant evidence, such as quoted by Mr. R. Price-Williams, that the higher-phosphorus rails, of the lower carbon specified, have given excellent service.

I am glad that Mr. Price-Williams, whose first paper on the life and wear of rails was presented to the Institution of Civil Engineers in 1866,[3] and who has since devoted much time to the study of this subject,[4] supports my criticism that some of the drop-tests of foreign rail-specifications are disproportionate to the force of impact that the rail receives in service.

Also, that he calls attention to the analyses quoted in his paper,[5] showing that the lower-carbon rails of the British specifications had withstood eight years' severe service, although they contained 0.10 per cent. of phosphorus; and to the analyses of the Great Northern British rails tested by Mr. W. G. Kirkaldy,[6] which were in straight line track for 24 years and had withstood a traffic of 57,000,000 tons; the hardest of these rails contained 0.38 per cent. of carbon and 0.126 per cent. of phosphorus.

The knowledge of these examples of good service given by

[3] *Proceedings of the Institution of Civil Engineers*, vol. xxv., pp. 353 to 428 (1866).

[4] *Proceedings of the Institution of Civil Engineers*, vol. xlvi., pp. 147 to 208 (1876); vol. cxxxvi., pp. 185 to 188 (1899).

[5] *Proceedings of the Institution of Civil Engineers*, vol. xlvi., p. 157 (1876).

[6] *Proceedings of the Institution of Civil Engineers*, vol. cxxxvi., p. 185 (1899).

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low-carbon rails of British specifications containing, however, between 0.08 and 0.10 per cent. of phosphorus, and many others which might be quoted, prompted my remarks on the phosphorus-content, and warranted me, in the standard specification recommended to cover rails to be rolled in America for export, in saying that the phosphorus could be 0.10 per cent. as a maximum, thereby permitting tenders to be made from all American rail-mills.

Although appreciating the force of Mr. Price-Williams's remark that, from the point of view of the railroad company, the matter of greatest importance is the longest wear without the danger of brittleness, this subject could not be touched on, because the paper was confined to a comparison of current rail-specifications, in which no requirements as to wear and life are included. A number of references to the wear of rails were, however, purposely included in the Bibliography supplementing the paper.

It was for this same reason that the important questions raised by Mr. Lamberton and Mr. Freir, and other questions, such as hammer-hardening, etc., could not be included within the scope of the paper.

It was also unfortunate that no reference could be made in the paper to the experimental use of rails of special composition, such as very high carbon basic open-hearth rails, or the alloy steels containing either nickel, chromium, or manganese, and the results of which trials I have watched with much interest.

Mr. Harbord's remarks on the different degrees of hardness in open-hearth and Bessemer steel of like carbon are pertinent and should be borne in mind in drafting individual specifications.

In answer to Mr. Hadfield's question as to the chemical composition of American rails withstanding certain drop-tests, I refer him to Table III. of my original paper, and for the drop-test and composition in the recommended specification to Tables VII. and VIII. It seems to me that the required impact-tests of Table VIII. form a safeguard against the brittleness feared from the recommended upper phosphorus limit of 0.10 per cent. in the steel.

In suggesting that I should have recommended 0.07 or 0.08

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per cent. of phosphorus so that the manufacturers' desired leeway will result in furnishing no steel of more than 0.10 per cent. of phosphorus, Mr. Stead overlooks the emphasis which I placed on the fact that the suggested standard specification must be strictly lived up to, and that the maximum allowable phosphorus must be 0.10 per cent. I am glad Mr. Stead speaks of high manganese as a frequent cause of brittleness. I have analyzed more than 200 broken rails in which I found the manganese to exceed 1.20 per cent.; in the recommended specification the highest allowable manganese, in the heaviest sections, is 1 per cent.

I am pleased to hear Mr. York, who has had so long a rolling-mill experience, lay such stress on the relation of section to finishing-temperature. Mr. Webster and Mr. Kenney criticise my omission, in the recommended standard specification, of clauses fixing a maximum shrinkage-allowance after hot-sawing, and a maximum camber for rails at the straightening-presses. Such restrictions are out of place in a specification recommended as a general standard for all shapes and weights of sections. They are justifiable as addenda to the specification when it is applied to a certain section, but even then some leeway must be given to allow for the difference in the method of rolling at each rail-mill. This answers Mr. Palmer's question about a definite specified shrinkage. Mr. Palmer overlooked clause (*f*), page 611, when he said that finishing-temperature was not touched on anywhere in the paper.

I am glad that Mr. Webster includes in his communication a concise review of the present main points of difference between the rail committees of three American societies, because, as I confined my comparisons to the requirements of the current standard and individual rail-specifications now governing the rolling of rails in England and the United States, I purposely omitted two of these specifications. The president, Mr. R. W. Hunt, in his brief remarks closing the oral discussion of my paper, observed that the report of one of these committees had not yet been accepted—that of the American Society of Civil Engineers; to this he might have added that the rail-specification of the American Railway Engineering and Maintenance of Way Association had not yet come into use.

I am of the opinion that the reason given by the engineer-in-

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chief of the Paris and Orleans Railroad for the omission of all chemical requirements from his rail-specification—namely, that accurate analyses, other than carbon, cannot be made without retarding manufacturing operations—would be warmly disputed by the chemists of French rail-mills. In American mills no difficulty is experienced, with an output of 2,200 tons of rails per 24 hr., in obtaining accurate determinations of carbon and manganese on each heat, phosphorus on six heats every 12 hr., and an average silicon and sulphur on each 12-hr. rolling.

Mr. Sauveur calls to account the makers of rail-specifications for not demanding that the impact-test shall always be made on a piece cut from a rail coming from the top of the ingot. In my recommended standard specification I leave the location of the test-piece an open matter to be settled between the inspector and the manufacturer. In my observation I have never seen any objection raised by American rail-makers, if the required height of drop is reasonable, to testing pieces of rail, or in practice usually rail-butts, cut from the top rail of an ingot. In July, 1905, at the eighth annual meeting of the American Society for Testing Materials,[7] at which Mr. Sauveur and many representatives of rail manufacturers were present, a standard rail-specification was adopted, in which it was specified that the drop-test shall be made on pieces from 4 to 6 ft., "taken from the top of the ingot."

Mr. Palmer raises the commercial question, often discussed, of the right of a manufacturer to invoice rails at actual weights when he has kept within the allowable variations from templet recognized as good rolling-mill practice. The allowable percentages of short rails and the number of second-quality rails for use at stations and in sidings and yards are also commercial matters for individual adjustment. The recommended specification, in these regards, was based on American practice.

A personal inspection of American mills furnishing rails for export would completely satisfy Mr. Palmer on the question of drop-tests and the requirements of the recommended specification as to check-analyses and inspection, for, as I have seen them interpreted by American mills, there is no just ground for complaint.

[7] *Proceedings of the American Society for Testing Materials*, vol. v., pp. 32, 43 and 47 (1905).

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Mr. Kenney refers to the testimony which has been furnished to the American Society of Civil Engineers and the American Railway Engineering and Maintenance of Way Association, showing that nearly every American railroad having heavy traffic is suffering greatly from broken rails. He states that this brittleness is due to phosphorus in the higher-carbon rails now ordered by American railroads, in their attempt to obtain increased wear under present heavy traffic conditions.

This criticism of 0.10 per cent. of phosphorus in the American high-carbon Bessemer steel rails does not apply to the maximum limit of 0.10 per cent. of phosphorus in the lower-carbon steel included in my recommended specification for rails rolled by American mills for export. I have referred elsewhere in this reply to Mr. Kenney's criticism, as to the absence in my recommended standard specification of a shrinkage-clause, and of a clause specifying a definite maximum camber for rails at the straightening-presses.

It was not from ignorance or from lack of appreciation, as assumed by Mr. Kenney, that I omitted references to the present American conditions touched on by him, but simply because my paper was confined principally to a criticism of foreign rail-specifications governing the manufacture of lower-carbon acid Bessemer steel.

Mr. Kenney declares that "the brittle rail is one of the most important subjects before American railroads to-day." He also says that "the rails (meaning American acid Bessemer rails) are not giving satisfaction as to wear; and any attempt to improve the wear by making the rails harder is met by a crop of brittle rails."

He remarks that this "brittleness is caused by high phosphorus." I think high phosphorus is one of the causes, but if it is a fact that the "brittle rail is one of the most important subjects before American railroads to-day," then the real cause of the trouble is that the railroads have continued to attempt to obtain better wear, by asking the makers of acid Bessemer rails to put more carbon in the steel, instead of benefiting by their own first experience with brittle rails and by the advice given them by the manufacturers, who are dependent on ores making acid Bessemer steel between 0.08 and 0.10 per cent. of phosphorus.

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Mr. Kenney asserts that rails with lower phosphorus and much higher carbon have shown much better wearing qualities without being brittle; he here refers to some experimental high-carbon basic open-hearth rails, now on trial. It will be some time before the ore-conditions of the United States are such that none of the American mills can make acid Bessemer steel rails under 0.10 per cent. of phosphorus. I venture to suggest that this interval of time can be advantageously spent by the railroads having heavy traffic in making experiments to improve the life of the rail, other than by simply increasing the carbon-content.

In a recent address[8] before the Merchants' Club of Chicago, Mr. James J. Hill, President of the Great Northern Railroad, in speaking of the railroad conditions in the United States, stated:

"I have noticed that from 1895 to 1905—ten years—the growth in ton-mileage was 110 per cent. The growth in the mileage of railroads to handle that traffic was 20 per cent. . . . In ten years the railroads of the country expanded 20 per cent. for the handling of a business that increased 110 per cent. . . . It is estimated that from 115,000 to 120,000 miles of track must be built at once to take care of this immense business."

During the ten-year period referred to by Mr. Hill the American railroads have not increased the sectional area or weight of their rails more than 25 per cent., and they have tried to make up this great deficit between immense increase in tonnage on one hand and small increase in trackage and in weight of rail on the other hand, by simply demanding acid Bessemer steel rails higher in carbon.

They have evidently, according to Mr. Kenney's statements about broken rails, passed the limit of safety.

In a letter which Mr. Hill sent to the Governor of Minnesota on Jan. 14, 1907, he gives the following exact data compiled from the official reports of the Interstate Commerce Commission, which further emphasize the greatly increased wear to which rails are now subjected:

	1895.	1905.	Increase
			Per Cent.
Freight ton-mileage	85,227,515,891	186,463,190,510	118
Passenger mileage	12,188,446,271	23,800,149,436	95
Total single-track mileage	180,667	218,101	Only 21
Number of locomotives	36,699	48,350	35
Number of passenger cars	33,112	40,713	23
Number of freight cars	1,196,119	1,731,409	45

[8] *Railroad Gazette*, vol. xli., p. 441 (1906).

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Mr. R. Price-Williams, an authority on the wear of rails, in his able discussion of my paper declares:

"There can be no question that the wear of steel rails subjected to the destructive effects of the great increase in the weight and speed of the main-line traffic of the principal railways in this country is far greater than is generally supposed."

He observes that the annual cost of maintenance and renewal of the London & North-Western was one-seventh of the entire working expenditure.

It certainly appears from these statements that the question of the maximum service safely obtainable from a rail must be looked upon broadly and studied seriously. The rail-maker must do his part; but the railroads, in view of the present serious traffic-conditions, must be willing to experiment on a large scale in order to determine whether a greater initial cost of the rail, by either increased section, special composition, or both, is not economy in the end.

Until a general change in American methods of rail-manufacture takes place, it appears to me, from the discussion, that the proposed standard specification will, if strictly lived up to, form the basis of a contract under which American mills will furnish a safe rail for export and one which will give as long a life in service as rails of British make.

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Piping and Segregation in Steel Ingots.

Discussion of the paper of Mr. Henry M. Howe, p. 3.

SECRETARY'S NOTE.—M. Beutter's introductory remarks, being simply a *résumé* of Prof. Howe's paper, with a disclaimer of the intention to criticise it, have been omitted, and the remainder of this contribution has been somewhat condensed, without omitting any essential statements.

M. BEUTTER, St. Etienne, France* (communication to the Secretary†):—Professor Howe observes in § 47 of his paper that the fluid-compression of steel diminishes piping by forcing the liquid and viscous metal, while the pipe is forming, or after it has formed, into the space so produced, and proceeds to study the different systems of compression, alluding to the four principal ones: Whitworth, Illingworth, Williams, and Harmet.

Having now for many years had in use the last-named process, that of M. Henri Harmet, I shall attempt to explain the difference between it and the three other processes named.

The Harmet process does not diminish or close the pipe after it has formed, but prevents it from forming; and I may add that it is the only one of the four which produces this result.

It is true that the Whitworth process diminishes the interior hollow by localizing it around the central axis of the ingot, which is afterwards removed by trepanning the hollow forgings for which this process has been, I believe, always exclusively used. The Illingworth and Williams processes, which, in my opinion, are applicable only to small ingots, of necessity allow the pipe to form, or (to use the expression of Professor Howe) they allow the surface of the lake of liquid steel to sink into the interior of the ingot, since they involve waiting a certain time before compression, so that (in the case of the Williams process) the ingot-mold can be opened without danger, or (in the case of the Illingworth process) so as to avoid the risk that the steel may enter the grooves produced by the removal of the distance-bars. During the time required for

* Chief Engineer at the Fonderies, Forges & Acieries de St. Etienne.

† Received Jan. 25, 1907, and translated by Franck E. D. Acland.

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the formation of the exterior skin of the ingot the level of the interior lake of steel has certainly sufficient opportunity to sink; and compression, ultimately applied, by either the Illingworth or the Williams process, can result only in raising more or less the level of this lake into the hollow parts already formed above it. But it is also certain that this liquid lake, as it ebbs from its banks, will have deposited upon it impurities, which may afterwards prevent the intimate combination of the mass with the new steel which will subsequently rise over them. Ingots so produced may possibly appear perfectly sound, but will certainly present a zone of weakness, corresponding, in the imagery of Professor Howe, to the banks of the lake of steel at its different levels, or the pocket which has been allowed to form and afterwards closed. Professor Howe clearly shows that the operation of compression ought to commence as soon as possible after casting, so as not to allow the level in this lake to fall and then again to be forced back—such a movement of the steel being clearly detrimental.

This evil does not exist in the Harmet process, which, as already explained, does not close up a pocket already formed, but prevents its formation by effecting the lateral compression of the metal immediately after casting. By reason of the conical form of the ingot-mold, it is not necessary to open the mold in order to commence operations, and consequently to wait until an exterior skin or crust is formed, during which delay an interior hollow has begun to be developed. In our own practice we have found that, with certain hard steels, it is practicable and advisable to begin compression by forcing the ingot through the mold while casting is still going on. In the steelworks at St. Etienne, an ingot is rejected as bad, if for any reason the operation of compression was not commenced upon it at once, even though it is possible that such an ingot might not show a pipe or pocket if a section were made of it. This point is illustrated by Fig. 1, representing the photograph of an ingot sectioned through its longitudinal axis. In order to produce this result we intentionally waited some time after the casting before commencing compression, doing our utmost to bring about conditions similar to those produced by the Illingworth and Williams processes, which cannot possibly commence compression at once. When sectioned, this ingot appeared to

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be perfectly sound; but, after a treatment with acid, the shape of the primary pipe, afterwards filled with metal forced into it by compression, was very clearly revealed. The photograph shows also that the whole axis of the ingot has been subjected to an upward movement of the liquated metal towards the pipe at the top. The axis of the ingot remains porous, and the treatment by acid reveals perfectly the manner in which the metal flowed.

FIG. 1.—SECTION OF AN INGOT, SHOWING EFFECT OF DELAY IN COMPRESSION.

Ingots compressed by the Harmet process, in which the pipe is never allowed to be produced, show, after treatment with acid, a remarkable solidity and compactness, without any trace of a zone of weakness. This Harmet process continually keeps the skin of the ingot absolutely full, and even slightly flowing out of the top.

The conclusion of Prof. Howe, that the Harmet is less effective than the Williams process, seems scarcely fair. His reason is, that the latter process squeezes the ingot at the center of its length only, thus allowing the metal to rise more readily, in order to fill the void above.

But since, in the Harmet process, the ingot is kept always full and compact, there must be an advantage in squeezing it uniformly and at all times over its whole surface, utilizing fully the wedge, due to the conical mold, and thereby energetically compressing the metal to its center, so long as there remains a single drop of liquid steel—thus producing, in consequence, ingots absolutely homogeneous and compact from center to surface, and from top to bottom.

Ingots produced by this process are not only physically sound, without any interior hollows, but also, to a remarkable degree, chemically homogeneous. Liquation is not produced, because the process furnishes the chief conditions specified by

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Prof. Howe himself as being necessary to prevent this phenomenon.

Professor Howe points out as the means to avoid liquation: (1) casting at low temperature; (2) casting small ingots; and (3) employing thick molds, not lined with sand, and preheated. The temperature at which steel is cast is, and should be, independent of the method of compression employed, but both the other conditions named are fulfilled in the Harmet process. In fact, thanks to the absolute soundness of the ingots, produced by tremendous squeezing in the passage of the metal through the conical mold, it is possible to produce from our compressed ingots forgings such as ordinarily require the casting of larger ingots, and thus to dispense with much of the heat-treatment and forging required to put into proper molecular condition ingots which have not been so compressed. Finally, the molds used in the process are necessarily thick, on account of the energy which they have to transmit. They cool very rapidly; and it is our common practice to assist this rapid cooling with water, which pours continuously down the mold and over the hoops which reinforce it. From both points of view—namely, that of piping and that of liquation—the Harmet process thus fills all the conditions desired.

On the other hand, the Whitworth process attacks the ingot in the direction of the length, in which it must present its greatest resistance; causes it to be distended outwards from the center, in a lined mold which does not assist the cooling; and results in producing ingots in which the axis is unsound, and which are used only for hollow forgings; whereas, the Harmet process attacks the ingot throughout its length with the whole lateral effort resulting from a conical mold, and, thanks to the thick iron mold, which cools very quickly, produces ingots sound and perfect throughout, and available, without waste, for forgings of the highest quality, such as armor-plates, guns, shells, and complicated cranks.

The foregoing statements and claims are not made upon theoretical grounds only, but can be verified by the records of daily practice at the steel-works of St. Etienne and other European establishments where thousands of ingots, widely known for their homogeneity, have been regularly manufactured by this process for many years past.

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HENRY M. HOWE, New York, N. Y. (communication to the Secretary*):—Mr. Beutter has completely missed the point of my discussion of the relative merits of the different methods of compression. I refer him to the first paragraph of § 62, which I think states the matter clearly, especially if taken in connection with §§ 60 and 61, pp. 91 to 93.

But perhaps I may recapitulate the matter by saying that what I was discussing was the efficacy with which we could make use of the pipe itself as a receptacle into which we could lift the segregate. I pointed out that in order that the pipe should be most effective for this purpose, we should leave the pipe itself uncompressed, so as to leave it open to receive as much as possible of the segregate. Compared with Williams's method, Harmet's has the disadvantage that it diminishes the pipe, not alone through lifting the segregate into it, but also by narrowing the upper part of the ingot, which contains the pipe, or at least which would contain the pipe were the pipe allowed to form. By as much as it thus narrows the upper part of the ingot, by so much does it diminish the size of the receptacle into which the segregate may be lifted, and by so much does it lessen our power to lift the segregate, and thereby to shorten that portion of the ingot which has to be rejected because of the segregate.

MR. ROBERT W. HUNT, Chicago, Ill.†:—I think the parties who have anything to do with commercial steel-products fully realize the very great importance of this subject. In fact, it is one that to-day is staring the railway world in the face with a vital importance. We read unfortunately often of railroad wrecks, which are attributed to broken rails. In a majority of cases, the rail is found to have been piped. This piping takes place to an extent much beyond what we appreciate or apprehend; for there are many thousands of rails in service which have not failed, and still have that trouble fully concealed in their interior; and these are a constant menace, being liable to fail at any time. Unfortunately, it means, in the present condition of the art, a revolution in rail-making, if this thing is met, as sooner or later it must be.

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* Received Feb. 14, 1907.

† Oral discussion at the New York Meeting, April, 1907.

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At present the element which seems to control is that of volume of production, over-controlling and over-topping every other consideration; and to say to the rail-maker that he must discard from 25 to 33⅓ per cent. of his ingots, is so revolutionary that it will not be listened to. But it seems to me this will have to be done, unless the difficulty can be overcome in some mechanical or chemical way. Prof. Howe, if I am not mistaken, suggests one or two ways in which this piping, and also the effects of segregation, can be minimized. But, if again I am not mistaken, he does not give us a remedy which would apply to rail-making. I think he suggests, for instance, that an ingot shall not be more than 8 in. in diameter. That, of course, is absolutely impracticable, so far as making rails is concerned. It would attack the vital condition of production, and it would give us a casting from which we could never roll successfully a 100-lb. rail, for instance.

This matter, as I have said, is of the greatest moment. Anything that tends to throw light on the existence of the disease must enforce consideration and help to bring about a remedy, even if it involves the cutting-down of production, and so an increased cost to the consumer of the manufactured rail. We cannot go on as we are doing now. We have mysterious breaks in which the rails do not show any piping, but break, apparently, without cause. Frequently, analysis discloses segregation, which perhaps sometimes accounts for the failure; but even that is not always present. The cause seems to my mind to date back to the physical manipulation of the steel during the process of manufacture.

We have gone away absolutely from the practices and traditions of the past; and the effort has been and is to get the metal into shape as quickly, and with as few applications of power, as possible. In the first place, the size of the ingot has been increased and the number of passes in the rolling decreased. If a structure, when cast in the ingot, is very coarsely crystalline, and in the finished product is much finer and more closely united—the units of crystallization having been reduced in size—it is to my mind illogical to assume that we can get the same result in that respect by forcing the change in crystallization immediately, as by giving it more time. We may effect some change in character, but is it not easy to appreciate that

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the physical properties and the strength of structure may differ? Whether we can realize it or not, it is true. We have had during the past few years an illustration which, to my mind, emphatically demonstrates it. There is a rail-mill and steel-works on this continent which, after becoming superannuated, and standing idle many years, was purchased and rehabilitated, and is now running. This plant is nearer the character of the works of 25 or 30 years ago than any other in existence on this side of the water. They re-melt the iron in cupolas; and the small size of the converters, the weakness of the blooming-mill, etc., necessitate a slow reduction of the ingots to blooms, and the re-heating of these before rolling them into rails; thus making the routine of manufacture closer to that of the olden time than at any other American works. A certain railroad has had in its track more than 100,000 tons of these rails, and quite an equal quantity of rails made by other manufacturers, more nearly in accordance with the present usual course of manufacture. The figures of breakages are in favor of the old-fashioned procedure, to the extent of quite four to one; although it happens also that the chemical composition of these rails that have broken less is not as good as that of the others. In fact, the phosphorus in them had a limit of 0.085 per cent. as compared with 0.07 per cent., or less, in the others.

Prof. Howe's paper, like everything else he has contributed, has been prepared with great care, and demonstrates very clearly what happens in casting steel ingots. It is up to the people who produce to use this knowledge and this light. For one, I want to thank him very much.

Everything has been ignored lately for speed. We must get back; and if mills cannot make 70,000 tons of rails a month, and make them right, somebody has got to force them back to the quantity which they can make and make right, or else let them keep them piled in their yards, and not put them in the track to endanger property, to say nothing of human life. The whole trouble is not with the rails. Road-beds, weight of rolling-stock, speed of trains, counterbalancing of engines—all must be considered; but we know of the rail-weakness, and its correction should not be delayed while the other elements of the situation are receiving attention.

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PROF. HENRY M. HOWE, New York, N. Y. (communication to the Secretary*):—These words of Captain Hunt, than whom none can speak with more knowledge and authority on the rail-question, are full of interest, especially his belief that the quality of the rails has been lowered by the increase in the size of the ingots and the decrease in the number of passes in the rail-mill.

As regards the remedies which I have suggested for segregation, I have had too much practical experience in rail-making to be caught making so rash a suggestion as decreasing the size of rail-ingots to 8 in. This size I advised for special cases of "steel of unusual excellence, such as high-carbon steel for the wires of suspension bridges." (§ 82 of my paper, p. 106.)

In order to restrain segregation in rail-ingots I would do three things: (1) use aluminum or its equivalent freely; (2) teem extremely slowly; and (3) increase the discard. In order to increase the practicable slowness of teeming of each ingot taken by itself, without running into the danger of having a heavy ladle-skull, I would teem into two or more molds simultaneously from the same ladle, either through two nozzles or through a distributing-funnel.

As to cropping, it is idle to crop off only the ragged top of the ingot, and leave just below it the richest of the segregate, the part into which the phosphorus has been concentrated from the whole ingot. Beyond this, I would have the inspectors search carefully for the segregate in a specified percentage of the ingots made; and I would give an upper limit for phosphorus in the most-segregated part, as explained in §§ 78 and 79 of my paper, pp. 103, 104.

I have expressed my views on the rail-question more fully in an article in the *Engineering and Mining Journal*, vol. lxxxiv., p. 21 (1907).

ALFRED C. LANE, Lansing, Mich. (communication to the Secretary †):—Prof. Howe's paper on Piping and Segregation in Steel Ingots, is, in my judgment, very interesting and suggestive to geologists, as well as metallurgists. For an igneous rock-mass is but an ingot, in the case of intrusives generally "bottom cast" (p. 59). Its general dimensions are also fixed

* Received June 19, 1907. † Received May 4, 1907.

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by those of a shell of frozen material, while most of it is hot and molten (pp. 8 and 12). The question whether rock-magmas contract or expand in solidification is also in doubt; and the same lines of argument apply (p. 9), but Howe's explanation (p. 12) is also applicable to the igneous rocks, since these likewise have a tendency to form cavities which may remain as vugs or druses or be filled up by pegmatites or aplites.

Prof. Howe's discussion of the location of the pipe is also applicable and the five factors which limit the same, though of course in the case of igneous rocks the pressure may be more than the atmospheric (p. 47). Blow-holes (pp. 37, 74) correspond somewhat to amygdules, and the ice-ingot bubbles (p. 75) to pipe-amygdules. The migration of metals below the melting-point (p. 48—also described by Dr. Koenig before the Lake Superior Mining Institute) is obviously important in the science of ore-deposits. The walls of the pipe, sometimes smooth and sometimes lined with crystals (pp. 50, 51, 54, 55), might also be matched among the cavities in igneous rocks. Druses of crystals, continuous with those making up the igneous rocks, have been described, for instance, by Patton. It is interesting to notice that there is sometimes (pp. 54, 55) a sort of transition-zone, well shown by the melilite slags of the copper-country, in which crystals occur with rounded faces, and heavily charged with cavities. The crystalline forces of the mineral have been able to distort the round shape of the drop somewhat, but not perfectly to free it from its mother-liquor. The "pine-tree" crystalline growths (p. 54) find frequent analogues in what are known as arborescent forms in rock-forming minerals.

The discussion of segregation, its causes and its restraining factors (p. 77), has an obvious application, almost word for word, to magmatic segregation. For instance, the segregation of the heavier augite in the ash-bed porphyrites is below the center, as in the ice ingot (p. 68). But when Prof. Howe comes to discuss the effect of the rate of cooling, I do not think that he has fully given due weight to the possible effects of the temperature of pouring relative to that of freezing; and it is along this line that I venture some suggestions which may have value for him and his readers.

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In the theory of cooling and its effect on grain which I have developed,[1] the essential points, other things being equal, are these: that the slower the cooling the coarser the grain; that the slowest rate of cooling at the time of freezing might be at the center or at the margin or somewhere between; that this depends most directly on the country-rock (or mold[2]), the distance to which the same is affected (whether sand or iron molds), the temperature of solidification (or freezing), and the initial temperature (or that of pouring). It may easily be that in considerably superheated magmas (ingots), or those in which the country-rock is already hot and a poor conductor (a mold already heated, or made of a non-conductor, such as sand), the locality of slowest cooling at the time of freezing will be somewhere near the margin. In fact, it may have any position, according to circumstances. Hence, the center or axis of the ingot may possibly be cooling much more rapidly than other parts, at the time of freezing. These relations are illustrated by curves of cooling, which I have elsewhere published. If this be true, those of Prof. Howe's statements which imply that the center must necessarily cool more slowly than the outer part (p. 13, § 7; p. 28, last paragraph; and a considerable part of the discussion of segregation from p. 77 on) must be qualified, or modified. It seems to me quite possible, for instance, that in the case of Fig. 28 (p. 74) the ring of blow-holes might be due to the fact that that was the ring of slowest cooling, and that, both at the margin and at the center, the cooling was too fast at the time of freezing to allow the gas to become disentangled. In the same way, it is possible that segregation may take place up to the distance from the margin which is that of the zone of slowest cooling, while, from this on, the more rapid cooling at the time of freezing might tend to check the continuance of this action, as is already brought out by Prof. Howe (pp. 82–85). This would account for some of the contradictory evidence cited by him (pp. 86–88). I have

[1] *Report of the Geological Survey of Michigan*, vol. vi. (1893-97); Fifth and Sixth *Annual Reports of the State Geologist of Michigan* (1903, 1904). See also *Bulletin of the Geological Society of America*, vol. xiv., pp. 369 to 406 (1902); *Journal of the Canadian Mining Institute*, vol. ix., pp. 210 to 217 (1906).

[2] In this passage, the phrases in parentheses are intended to indicate the conditions and factors considered by Prof. Howe, which correspond to those enumerated by me, in discussing the analogous geological problems.

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called attention to possible cases of this character in the paper for the Canadian Institute.[3] The essential thing to remark is, that in considering such evidence as that of the temperature of pouring (§57, p. 86), the superfusion ratio is a most important factor, without knowledge of which it is hardly possible to discuss the data to advantage.

PROF. HENRY M. HOWE, New York, N. Y. (communication to the Secretary*):—The points of likeness to which Dr. Lane calls attention, between the solidification of metallic ingots and that of rock-masses, are full of interest. I have elsewhere examined these resemblances,[4] and in my lectures I have always insisted on them, and found them very effective in fixing the students' attention.

I am at a loss to understand how Dr. Lane can have thought that I asserted or implied that the cooling of the center in the act of freezing is slower than that of the outside. Why, the very essence of my explanation of the formation of the pipe is that the central parts, when they are freezing, are cooling faster than the outer parts. (§§ 9, 10 and 11, pp. 14–16.)

In their solidification, steel ingots differ in one very important respect from rock-masses—viz., that the steel ingot is cast when only very slightly above its freezing-point into a mold which is habitually cold, so that the outer layers begin to freeze almost instantly, and that, at the beginning of the solidification, their cooling must be much faster than that of the central parts. I quite agree with Dr. Lane that, when the central parts in turn come to freeze, they may be cooling faster or they may be cooling more slowly than the outer parts. If they are cooling faster, a pipe should form; if more slowly than the outer parts, the pipe should be absent, as it very often is.

What I was trying to do was to explain how it comes about that the pipe forms, and I pictured the conditions which lead to its formation, among them the faster cooling of the center than of the outside at the time when the center is freezing. I by no means intended to imply that this pipe-forming condition, which is essential to pipe-forming, must always exist. I

[3] *Journal of the Canadian Mining Institute*, vol. ix., pp. 210 to 217 (1906).

* Received July 26, 1907.

[4] *Iron, Steel and Other Alloys*, pp. 2 to 8.

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might indeed have done well to affirm that it need not, though that would have been foreign to the argument before us.

When we turn to segregation, the important matter is not the rate of cooling of the central parts at the time of their freezing, but the fact that their freezing occurs later than that of the outer parts; not rapidity, but priority.

ALFRED C. LANE, Lansing, Mich. (communication to the Secretary*):—The misunderstanding of the implication of certain sentences of Professor Howe's paper is not so much to be regretted, since it has drawn out the valuable and clear supplementary statement. I did not know enough about the technique of steel-ingot casting to take for granted that the ingot is "cast when only very slightly above its freezing-point," but had supposed that the amount of excess of temperature or superfusion might be, as it is competent to be, an important factor.

Laboratory Experiments in Lime-Roasting a Galena-Concentrate.

Discussion of the Paper of H. O. Hofman, R. P. Reynolds and A. E. Wells, p. 126.

GEORGE A. PACKARD, Boston, Mass. (communication to the Secretary†):—The very interesting results obtained by Prof. Hofman and his assistants came to my attention when I had temporarily assumed charge of the metallurgical department of the Missouri School of Mines, and had found it, pending the completion of a new building, lacking a furnace in which to roast ore preparatory to the customary lead blast-furnace run. An attempt at lime-roasting had been already determined upon, but the use of a charge which would make a slag of definite composition was a new idea. The results following were obtained by Messrs. Aubrey Fellows, J. C. Long, E. R. Wash, and Dr. T. C. Tseung, under my supervision. It is regretted that the data are not more complete; but the work was done simply incidentally to securing the desired ore for the blast-

* Received Aug. 23, 1907. † Received Apr. 12, 1907.

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furnace, with no idea of obtaining any results of especial interest. It is submitted because of the metallic lead obtained, no notice of which has been seen elsewhere.

The ore was an aggregate of several lots of concentrates, apparently from Missouri ores. The analysis follows, with that of the limestone and sandstone used, as far as made: Ore, SiO_2, 3.04; Fe, 0.77; CaO, 2.53; Pb, 68.74; Zn, 5.77; S, 13.9 per cent.; limestone, SiO_2, 1.0; CaO, 54.35 per cent.; sandstone, SiO_2, 99.8 per cent.

The converter was made in the shops from a sheet of No. 14 iron riveted in the shape of a truncated cone, 20 in. in diameter at the top, 15 in. at the bottom and 22 in. high. A grate of No. 8 iron, 17 in. in diameter, with 130 $\frac{3}{8}$-in. holes placed at about 1.25-in. centers, was held in place by the shape of the converter about 4 in. above the bottom. This converter was set over a low brick flue, 9 in. square, into the bottom of which air was delivered by a 1.5-in. air-pipe. After the first run the air was supplied by a Laidlaw-Dunn-Gordon compressor having an 8-in. stroke and a 13-in. cylinder, running 120 to 132 rev. per min. All of this air passed through the ore excepting what a 0.5-in. pipe would deliver to the hood to carry off the fumes by forced draft.

The arrangements for starting the roast were the same as described by Prof. Hofman, excepting that coke was used in place of charcoal in the first run.

The first charge was made up of 229 lb. of ore with 20 per cent. of limestone and 20 lb. of silica, giving a silicate degree of oxygen of base to acid of 1.6 to 1. Ore, lime, and silica were mostly of 3-mm. size, but all contained perhaps 10 per cent. of larger-sized slab-shaped pieces which had passed the rolls set to $\frac{1}{8}$ in. Coke was used in place of charcoal for starting the reaction. In this case the air was supplied, through a connection with a furnace-tuyere, from a No. 1 Baker blower, and not over 4 in. of water-pressure was obtainable.

Sulphurous fumes were evolved slowly and the total time was about 2 hr., though but little sulphur was noticeable after 35 min. Examination of the resulting mass showed it to be only partly sintered and but 15 per cent. of the sulphur eliminated. Evidently finer crushing, more blast, or some other change was necessary. After picking off all the adhering coke

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this whole mass was now re-crushed to 3-mm. size, and again charged into the converter, the compressor supplying air for the second run. The only difference in composition between this and No. 1 was the small amount of sulphur expelled, and an increase of perhaps 50 per cent. in the limestone from that used under the first charge which adhered to the sintered mass and was re-crushed with it. The result showed the sulphur in the charge reduced to 2.6 per cent., but a large amount of metallic lead was obtained. This was not separated, but was estimated to be equal to at least 10 per cent. of the total lead in the charge.

A series of experiments was now undertaken with a view to obviating this difficulty. While it is possible that the lead might be reduced by the action of lime, it seemed probable that owing to the small amount of silica present, about 8 per cent., the reactions were those of the "roast and reaction process," the lead sulphate and oxide reacting with the sulphide to give metallic lead. These experiments were carried on in a clay crucible similar to that used by Prof. Hofman, and 1 kg. was the unit of ore used.

Series A was run with low silica, about 9 per cent. of the charge, and 40 per cent. of limestone, having an oxygen ratio of base to acid of 1.5 to 1. The pressure varied from 2 in. to 8 in. of water. The amount of lead obtained in the metallic form on a 100-mesh sieve varied from 0.11 to 2.85 per cent. of the total lead present, the highest with the lowest pressure.

Series B was run with low silica, about 9 per cent. of the charge, but only 20 per cent. of limestone added, giving approximately a singulo-silicate ratio when the lime in the ore is included. The result was, from 1.43 to 3.03 per cent. of the total lead was obtained in metallic form, but in this case the highest per cent. was obtained with the highest pressure.

Series C was run with high silica, 17 per cent. of sandstone added, and enough lime to give a singulo-silicate ratio, including the silica and lime in the ore. In this series no metallic lead was obtained.

A single additional experiment, with 12 per cent. of sandstone added and limestone proportioned as in series C, gave 0.39 per cent. of metallic lead on a 100-mesh screen.

The amount of sulphur eliminated varied in the tests from

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48.8 to 75.5 per cent., though it was below 61 per cent. in only two out of ten runs. Our results in this connection seem too variable to serve as a basis for conclusions as to the effect of varying pressure, especially in the cases where metallic lead was obtained. The same condition seems to exist in the results published by Prof. Hofman. Thus 4, 4a and 4b, using 10-, 2- and 10-in. pressure respectively, show less variation between the first 10-in. pressure and the 2-in. than between the two having the same pressure. In our series, with no lead formed and pressure being 3.6, 6 and 9 in., we obtained an elimination of 61.9, 65.9 and 73.8 per cent. of sulphur, but working in these small crucibles it seems possible the amount of charcoal present at the start might affect results. We might also add that the desulphurization was quite as good in series C with high lime as in B with low lime. But here again we have another variable, as the silica was also increased.

To us, the most practical result of these tests was the elimination of the metallic lead, a result which was confirmed by three subsequent 300-lb. charges in the sheet-iron converter, using 15 per cent. of silica in the charge.

Our conclusions were, that with this ore, with less than 12 per cent. of silica present, a considerable amount of metallic lead might be obtained; and also that our results on this ore did not confirm the results obtained by Prof. Hofman on the Cœur d'Alene ore as to the percentage of lime and the pressure giving the best desulphurization. Whether in working on ores very low in silica the amount of metallic lead might be increased to a point where this lead would be a product worthy of separation would be interesting to know.

H. O. Hofman, Boston, Mass. (communication to the Secretary*):—The conclusions arrived at by Mr. G. A. Packard in his experiments of lime-roasting a galena-concentrate containing SiO_2, 3; and Pb, 69 per cent., bring out several interesting points.

In showing that it requires the presence of 12 per cent. of silica to prevent the liberation of metallic lead, he calls to mind a similarity between the results of lime-roasting and slag-roast-

* Received May 8, 1907.

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ing the same class of ore in a hand-reverberatory roasting-furnace.

When I was connected with the smelting-plant at Mine La Motte, Mo., in 1881, it was necessary not only to add enough silica to the galena-concentrate to obtain a roasted product with 15 per cent. of silica, if the separation of lead was to be avoided, but to give special attention to a careful mixing of ore and sand, and to a thorough rabbling while the charge was passing through the furnace, as some lead would separate out in spite of the large addition of sand, if these two points were not attended to. An examination of the older analyses of slag-roasted ore and of the more recent ones of ore that has been only agglomerated reveals the fact that the quantity of silica rarely goes below 15 per cent.

This intimate contact of silica and galena being essential for satisfactory work in the hand-reverberatory furnace, furnishes one important reason, of the many that are possible, why the results in lime-roasting a galena, with 6.5 per cent. of gangue, of which 3 per cent. is silica, when mixed with sand may differ from those of a galena-concentrate containing 33 per cent. of acid gangue, of which 18.6 per cent. is silica.

The idea, suggested by Mr. Packard, of lime-roasting a galena-concentrate very low in silica without the addition of siliceous matter, with the object of recovering part of the lead in the metallic state, does not appear to offer any advantages on account of the inherent loss of lead by volatilization, the imperfect elimination of sulphur, and the difficulty of recovering the metal.

As soon as metallic lead separates out in slag-roasting, the fumes passing off by the chimney show this at once. The loss is sufficiently large to have prompted the uniform practice of avoiding it. Something similar will occur in lime-roasting. Even supposing the separation of lead to be permissible with non-argentiferous galena, it would be inadmissible when the galena carried any silver, as the loss in precious metal would be too great.

In lime-roasting, the charge becomes scorified; and scorified material gives an imperfect reduction of lead oxide and lead sulphate by lead sulphide; hence the elimination of sulphur by the operation would be unsatisfactory.

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There is, finally, the difficulty of separating the metallic lead from the lime-roasted ore. While some of it may trickle down through the charge, and, after passing through the openings in the grate, may be removed from the bottom of the converting-vessel, the bulk will remain disseminated through the roasted material. As this has to be smelted in the blast-furnace, there will be a decided loss in lead when the charge descends in the furnace, which everybody has experienced who has been forced to add bars of lead to his smelting-mixture in order to keep hot the lead in the crucible.

Relative Elimination of Iron, Sulphur, and Arsenic in Bessemerizing Copper-Mattes.

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PROF. HENRY M. HOWE, New York, N. Y. (communication to the Secretary*):—The results presented in Mr. Mathewson's paper are of very great interest. Perhaps the most interesting is that shown in Table I. and Fig. 1, in which the arsenic decreases in 10 min. from 0.22 to 0.07 per cent. This is such a large decrease that it is to be hoped that Mr. Mathewson will verify it by repeating his tests with a matte of similar composition. This is particularly to be desired, because the simultaneous removal of sulphur appears to be very small. All three blows further agree in showing that the removal of arsenic is in the very early part of the blow. The fact that the removal of arsenic then ceases abruptly cannot be explained completely by the fact that the arsenic has fallen so low as by its lowness to resist further removal, because in the first blow the arsenic becomes stationary at 0.06 per cent., whereas in the third blow it does not become stationary until it has fallen to 0.034 per cent. The abrupt ending of the removal of arsenic is more likely to find its explanation in the composition of the slag, and it is to be hoped that Mr. Mathewson will supplement these extremely valuable data with others in which the composition of the slag at the same time shall be given.

The sudden rise of arsenic toward the end of the third blow I suppose must be referred to an error in sampling or analysis.

* Received Feb. 9, 1907.

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[NOTE.—In this Index the names of authors of papers are printed in small capitals, and the titles of papers in italics. References to papers expressly treating of the subject named are likewise in italics; and casual notices, giving but little information, are usually indicated by bracketed page-numbers. The titles of papers presented, but not printed in this volume, are followed by bracketed page-numbers only.]

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ERRATA.

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406	2	For "Aug. 23" read "Aug. 22."
409	25	For "December, 1902," read "Apr. 24, 1902."
811	Last.	For "vanadine" read "vanadium."
813	First.	For "Vanadine" read "Vanadium."

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